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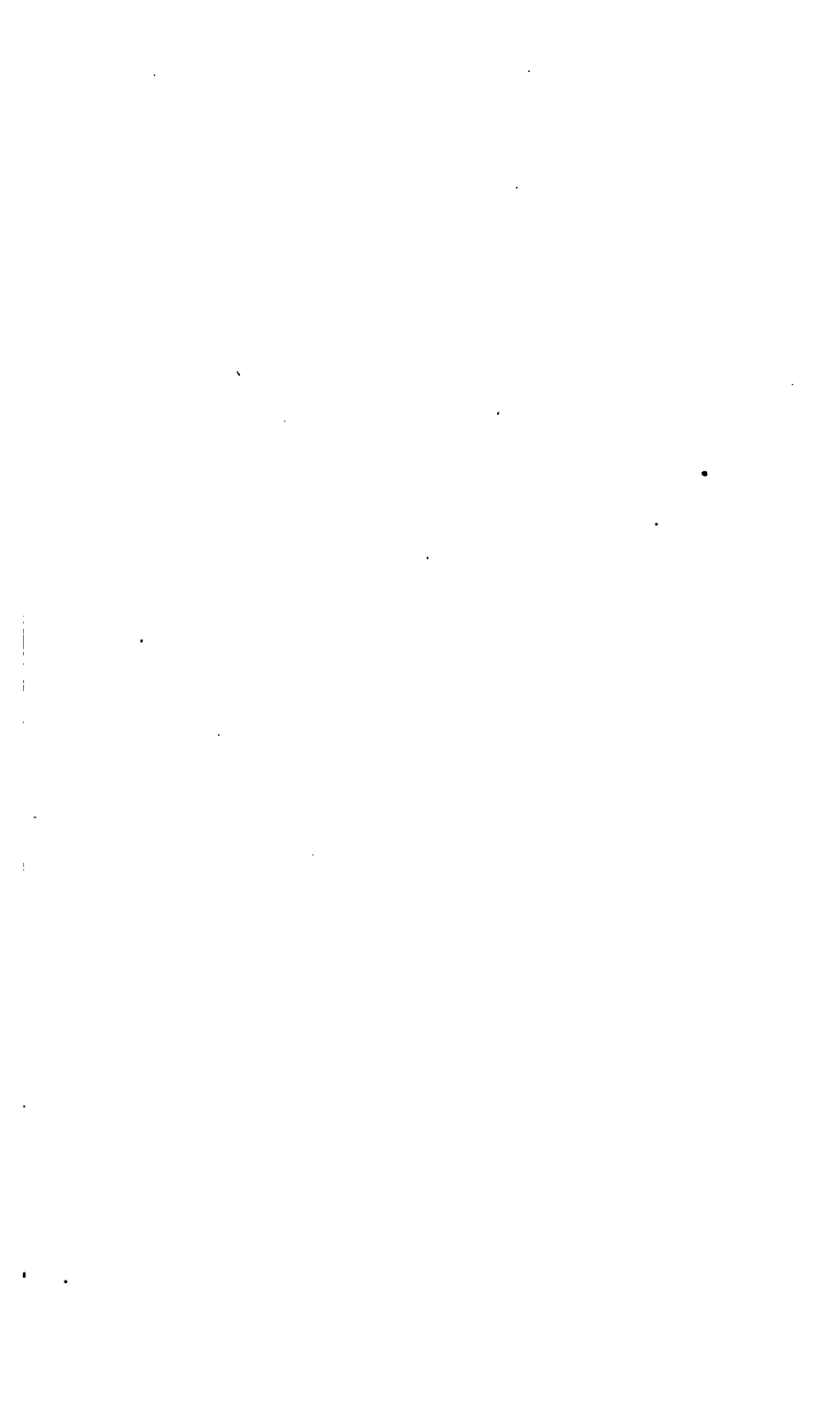
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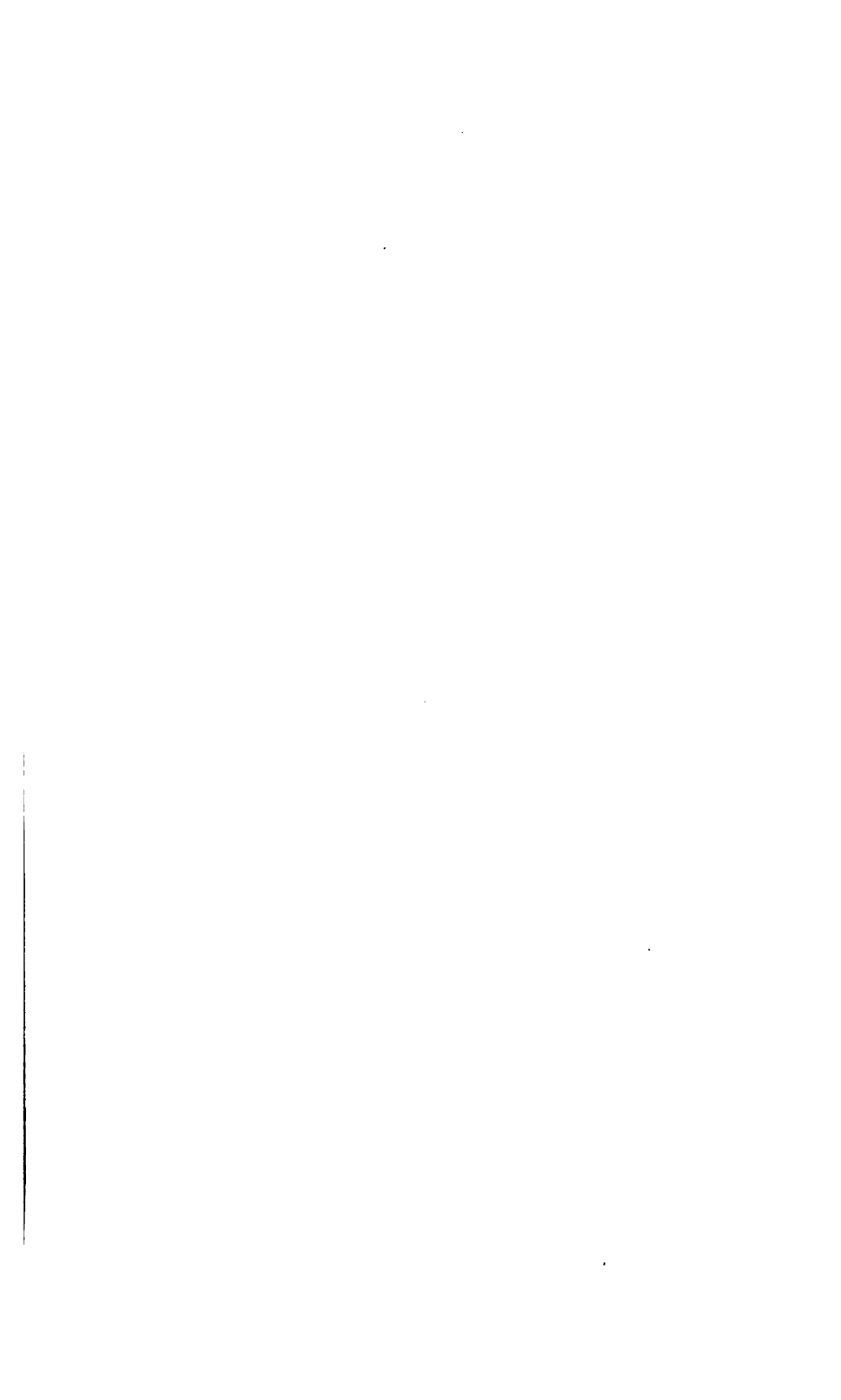


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Bulletin 133

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

**THE WET THIOPEN PROCESS FOR RECOVERING
SULPHUR FROM SULPHUR DIOXIDE
IN SMELTER GASES
A CRITICAL STUDY**

BY

A. E. WELLS



**WASHINGTON
GOVERNMENT PRINTING OFFICE
1917**

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THE WET THIOPEN PROCESS FOR RECOVERING SULPHUR FROM SULPHUR DIOXIDE IN SMELTER GASES; A CRITICAL STUDY.

By A. E. WELLS.

INTRODUCTION.

A critical study of the wet Thiopen process for extracting sulphur from the smoke of smelters is one of several investigations related to the general smelter-smoke problem that are being conducted under the direction of the Bureau of Mines. Part of the work done in the pyrometallurgical section of the cooperative metallurgical exhibit of the Bureau of Mines at the Panama-Pacific International Exposition during 1915 related to this investigation, and at the expiration of the exposition, the work was continued in the laboratories of the Bureau of Mines in the Hearst Memorial Mining Building of the University of California, Berkeley, Cal., where a mining experiment station of the bureau has been established under the cooperative agreement with the University of California.

Early in the spring of 1915, when plans were being made to cooperate with several metallurgical interests, the Thiopen Co. proposed to the Bureau of Mines that, along with the studies of various proposed methods for eliminating sulphur dioxide from smelter smoke, there should be included a study of the wet Thiopen process. In the study as outlined it was proposed to determine whether it would be technically and commercially feasible to use the process for the recovery of elemental sulphur from the sulphur dioxide in waste smelter gases.

Although there had been no large-scale work with the process, either in the laboratory or at a smelter, representatives of the metallurgical and mining industry had shown considerable interest in the proposed method, and the desire had been expressed that a better knowledge of the possibilities of the process should be obtained. Thus, cooperation with the Thiopen Co. seemed justified by the interests of the metallurgical industry, and by the conservation promised by the recovery of a product of market value from what is otherwise a waste, and in some instances a serious nuisance.



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No attempt was made to scrub the gases completely of their entrained solid matter. However, most of the dust settled, and as the material being roasted was nearly pure pyrite, there was little metallic fume carried over into the absorption tower.

Velocity determinations and temperature readings were made and samples of gases for analyses were taken at the fan, on the suction side.

Except during the tests when water was used as the absorbing liquid, the main object of the work with the absorption tower was to

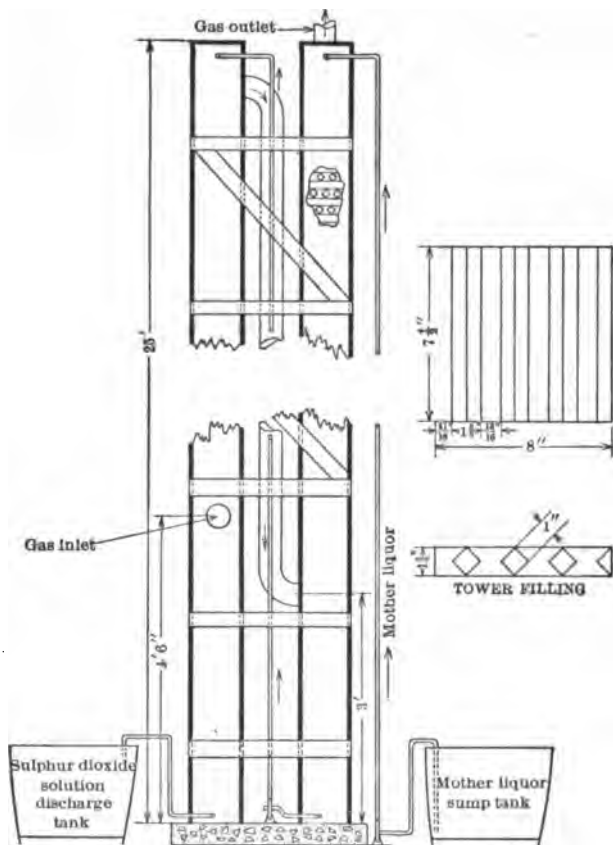
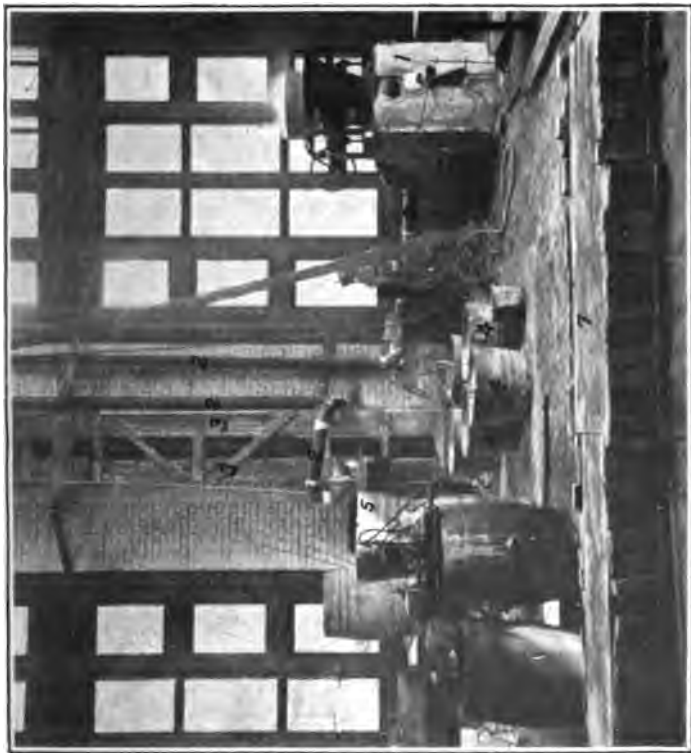


FIGURE 1.—Elevation and details of absorption tower used in experiments.

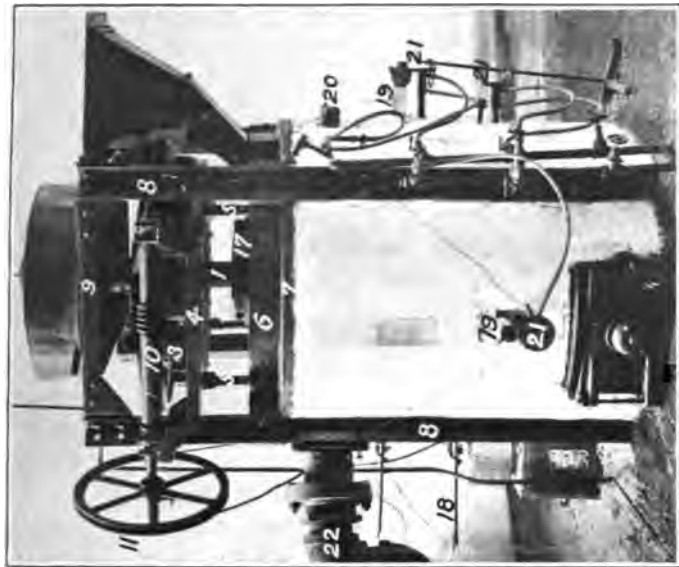
obtain a sulphurous-acid solution for use in precipitation. However, in all tests in which it was possible, the following records were taken, nearly simultaneously, in order to indicate the efficiency of the tower for absorption purposes:

1. Volume and SO_2 content of gases to tower.
2. Volume, temperature, and SO_2 content of solution from tower.

In certain tests, the SO_2 content of the exit gases from the tower was also determined.



A. APPARATUS FOR STUDY OF WET THIOGEN PROCESS: 1, MULTIPLE-HEARTH ROASTING FURNACE; 2, PIPES FOR COOLING GASES; 3, ABSORPTION TOWER; 4, PRECIPITATION TUBS; 5, FILTER TUBS; 6, SUCTION PUMP; 7, DRYING PLATES.



B. MULTIPLE-HEARTH ROASTING FURNACE USED IN TESTS.

ABSORPTION OF SULPHUR DIOXIDE IN WATER.

In October, 1915, at the exposition laboratory, tests were made to determine the maximum concentration of sulphur dioxide that could be obtained when water was used as an absorber, corresponding to different concentrations of sulphur dioxide in the gases.

In the tests the total volume of gases going to the tower was varied between 20 cubic feet (566 liters) per minute and 30 cubic feet (850 liters) per minute, the velocity of the gases through the tower thus varying between 1.5 feet (45.8 centimeters) per second and 2.2 feet (67.1 centimeters) per second. The average velocity was 2.0 feet (61 centimeters) per second. The volume of water passing through the tower was varied between 4 liters (1.05 gallons) and 15.5 liters (4.1 gallons) per minute. With a volume of about $8\frac{1}{2}$ liters per minute being pumped into the top of the tower, the total volume of water in the tower at any one time was about 20 liters (5.3 gallons). The water temperature varied between 15° and 19° C.

When the volume of water descending through the tower was less than 8 liters per minute (about 2.1 gallons), the concentration of the sulphur dioxide in the solution was close to that of theoretical saturation for water at those temperatures in equilibrium with the gas concentrations. The solubility of sulphur dioxide in water as given by Schönfeld^a is taken as the basis for calculating the theoretical saturation concentration. For example, with an SO_2 content of 6.5 per cent in the gases in water at 18° C., a maximum concentration of about 7.4 grams per liter (0.99 ounce per gallon), was obtained, as against a theoretical saturation concentration of 7.8 grams per liter.

With an SO_2 content of 8.5 per cent in the gases the water solution at 18° C. contained 9.8 grams of SO_2 per liter, as against a maximum saturation concentration of 10.2 grams per liter.

However, in these tests in which nearly theoretically saturated solutions were obtained, not more than 50 per cent of the total SO_2 going to the towers was recovered. (See Table 1.)

By increasing the volume of water going through the tower to 11 liters per minute, about 70 per cent of the total SO_2 was recovered. The concentration of the SO_2 in the resulting solutions was within 80 per cent of that for the theoretical saturation. (See Table 1.)

In order to increase still further the total absorption of the sulphur dioxide, the volume of the solution descending through the tower was increased to as much as 16.5 liters (4.4 gallons) per minute. With this large volume of water the total absorption of the SO_2 was increased to as high as 90 per cent, but the concentration of the

^a Schönfeld, Franz, Ueber den Absorptions-coefficienten der schwefeligen Säure, des Chlors und des Schwefelwasserstoffs: Liebig's Ann., Bd. 95, 1855, p. 1; see also Landolt-Börnstein Physikalisch-Chemische Tabellen, 4th ed., 1212, p. 509.

sulphur dioxide in these solutions was in some cases as low as 70 per cent of that required for saturation. (See Table 1.)

Table 1 following gives a summary of the results of the tests.

TABLE 1.—*Summary of tests of absorption of sulphur dioxide in water.^a*

[Dates of tests, Oct. 22-27, 1915.]

SOLUTION NEARLY SATURATED.

	Number of tests averaged.	Volume of gases to tower at 18° C. and 760 mm.	Temperature of gases at tower.	Per cent of SO ₂ in gases to tower, average.	Weight of SO ₂ to tower per minute, average.	Volume of water through tower, per minute.	Temperature of water at outlet.	SO ₂ content of solution.		Per cent of theoretical saturation.
								Per liter, average.	Total per minute, average.	
1915.		<i>Cu. ft.</i>	<i>°C.</i>		<i>Grams.</i>	<i>Liters.</i>	<i>°C.</i>	<i>Grams.</i>	<i>Grams.</i>	
Oct. 22.....	4	26	20	3.7	72.5	8.0	17	4.5	36.0	98
21.....	7	20	21	5.4	87.5	7.5	19	6.0	45.0	95
23.....	4	21	21	6.3	102.0	7.6	19	7.0	53.2	93
23.....	11	24	22	7.8	139.0	8.0	18	8.6	70.4	94
23.....	8	22	22	9.4	152.0	8.0	19	10.0	80.0	92

SOLUTION LESS SATURATED, BUT HIGHER PERCENTAGE OF SO₂ RECOVERED (APPROXIMATELY 70 PER CENT).

Oct. 22.....	5	25	20	2.5	68.6	12.0	17	4.0	48.0	91
27.....	8	22	22	6.0	97.2	11.0	18	6.2	68.3	86
23.....	3	21	21	7.0	113.5	11.0	18	7.1	78.2	84

UNSATURATED SOLUTION, RECOVERY OF SO₂ MORE NEARLY COMPLETE (APPROXIMATELY 85 PER CENT).

Oct. 22.....	5	21	22	5.2	84.2	16.2	17	4.8	77.8	74
22.....	4	21	20	6.5	105.2	15.5	17	5.7	88.5	71
27.....	9	22	21	7.2	116.8	15.0	18	6.1	91.6	71
27.....	3	21	22	8.1	131.0	16.0	18	7.0	112.0	72

^a Tests by A. E. Wells, W. Freeman, R. Buhman, C. E. Brandt, and A. L. Tuttle.
^b Grams per liter multiplied by 0.133—ounces per gallon.

ABSORPTION OF SULPHUR DIOXIDE IN MOTHER-LIQUOR SOLUTIONS.

The composition of mother liquor circulated through the tower to absorb the sulphur dioxide was not constant. The amount of barium in solution varied, according to the conditions under which the preceding precipitation had taken place, between 2 and 15 grams per liter. The ratio of sulphur to barium varied between 2 atoms of S to 1 atom of Ba, and 5+ atoms of S to 1 atom of Ba. These data, summarized briefly below, were obtained with a mother liquor having an average specific gravity of 1.035, a barium content of about 10 grams per liter, and a ratio of sulphur to barium of 3 to 5 atoms of S to 1 atom of Ba.

In determining the sulphur dioxide present in these solutions, samples were taken from the stream running out of the tower. These were titrated with a standard alkali, methyl orange being used as an

indicator. With methyl orange, neither the thiosulphate, the thionates, nor the bisulphite that might be present in the solution was titrated, and the alkali used was equivalent only to one-half the free sulphur dioxide. In other samples the total iodine value of the sample was determined. The samples were then heated to 70° to 80° C. to drive off the sulphur dioxide, and the iodine equivalent of the thiosulphate or bisulphite was determined, this equivalent being deducted from the total iodine value.

In 56 tests, with the velocity of the gases through the tower averaging 2 feet per second, and with the sulphur dioxide content varying between 4.8 and 6.5 per cent, approximately 73 per cent of the total sulphur dioxide was recovered with an average volume of 9 liters (2.37 gallons) of solution per minute descending through the tower. The resulting concentrations of sulphur dioxide in the solutions averaged 20 per cent greater than those theoretically possible in pure water at the same temperature, and in equilibrium with the gas concentrations.

TABLE 2.—*Absorption of sulphur dioxide in mother liquor of Thiogen process, with SO₂ content of 4.8 to 6.5 per cent.^a*

[Dates of tests, Oct. 11–Nov. 26, 1915.]

Number of tests.	Volume of gases to tower at 18° C. and 760 mm.	Temperature of gases.	Per cent of SO ₂ in gases to tower, average.	Weight of SO ₂ to tower per minute, average.	Volume of solution through tower per minute.	Temperature of solution at outlet.	SO ₂ content of mother liquor.		Per cent of total SO ₂ absorbed in mother liquor solution.	Ratio of concentration percentage attained to theoretical saturation concentration in water
							Per liter.	Total in solution.		
	Cu. ft.	° C.		Grams.	Liters.	° C.	Grams.	Grams.		
6.....	20	21	5.4	82.0	8.0	20	7.4	59.2	72	122
8.....	23	20	5.1	89.0	4.1	15	8.6	35.2	40	125
6.....	25	19	5.0	94.8	10.0	18	7.2	72.0	76	119
10.....	26	17	4.8	94.3	11.2	17	7.2	80.6	85	120
4.....	28	18	6.0	127.0	8.2	17	9.0	73.8	58	120
4.....	26	17	6.3	123.0	11.7	20	8.65	101.0	82	121
4.....	26	18	4.8	94.3	10.2	22	6.6	67.4	72	130
4.....	23	19	5.5	96.0	10.0	15	8.9	89.0	93	119
4.....	23	19	6.5	113.0	7.8	17	11.2	87.4	77	137
6.....	23	19	6.5	113.0	10.0	22	8.0	80.0	71	117

^a Tests by A. E. Wells, W. Freeman, C. E. Brandt, A. L. Tuttle, and R. Buhman.

Thirty-two tests were made with the sulphur dioxide content of the gases varying between 7 and 10 per cent, and the volume of solution used varying between 7.0 and 7.5 liters per minute. The sulphur dioxide concentration of these solutions averaged about 12 per cent greater than was theoretically possible in pure water.

Ten tests were made when the gas concentration varied between 11 and 13 per cent sulphur dioxide, the volume of mother liquor circulated being about 7.5 liters (2 gallons) per minute. In these solutions

the sulphur dioxide concentration was only about 8 per cent greater than that for water saturation under the same conditions.

TABLE 3.—*Absorption of sulphur dioxide in mother liquor of Thiopen process, SO₂ content of gases above 7 per cent.^a*

[Dates of tests, Jan. 29 and Feb. 1, 1916.]

Number of tests.	Per cent of SO ₂ in gases.	Volume of solution used per minute.	Temperature of absorbing solution from tower.	SO ₂ content of solution per liter.	Ratio of concentration attained to theoretical saturation concentration in water.
		<i>Liters.</i>	<i>° C.</i>	<i>Grams.</i>	
8.....	7.1	7.2	23	7.6	107
2.....	7.6	7.2	27	7.2	110
18.....	7.9	7.6	22	9.2	111
4.....	9.1	7.0	19	12.4	117
6.....	9.6	7.2	23	10.9	115
6.....	9.6	7.1	19	12.7	113
5.....	9.7	7.1	23	11.0	113
4.....	10.7	7.2	27	10.0	108
1.....	12.4	7.0	24	12.4	104
1.....	12.5	7.1	24	12.8	108
3.....	12.8	7.0	28	11.8	110
1.....	12.9	7.1	24	13.5	110
4.....	15.2	7.1	27	13.3	100

^a Tests by A. E. Wells, W. Freeman, V. P. Edwards, and R. Buhman. ^b Average—112. ^c Average—108.

It is, of course, hardly true that the solution factors obtained in the tests represented actual saturation. However, inasmuch as with pure water the theoretical maximum concentration within 10 per cent was obtained, it is safe to assume that under the same conditions the tests with the mother liquor gave results that came within that limit.

These figures are slightly lower, about 10 per cent, than those obtained in a small glass absorption tower by D. E. Fogg,^a former engineer for the Thiopen Co., who found also a decrease in the absorption coefficient with the higher gas concentrations, similar to that found in these tests. A comparison of concentrations obtained at 18° C. is given below.

Comparison of concentrations obtained in absorption tests.

Per cent of SO ₂ in gases.	SO ₂ concentration per liter in mother liquor with a specific gravity of 1.03.	
	Tests by D. E. Fogg.	Bureau of Mines tests.
	<i>Grams.</i>	<i>Grams.</i>
5	8.0	7.2
6	9.4	8.7
7	10.7	9.7
8	11.8	10.9
9	13.0	12.1

The results of absorption tests indicate that by the use of mother liquor as the absorbing liquid the concentration of the sulphur dioxide

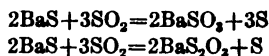
^a Figures given in personal communication to author.

in the solution can be brought to at least that of a saturated solution of water under the same conditions and with the usual gas concentration found in practice the concentration will be 10 to 20 per cent higher than is theoretically possible in pure water. By the use of a tower of the same type and height as used in the experiments and with the gas velocity varying between 2.0 and 2.5 feet per second, these concentrations can be obtained simultaneously with the recovery of about 75 per cent of the sulphur dioxide in the gases.

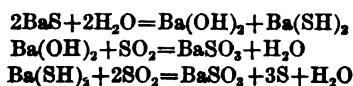
PRECIPITATION OF SULPHUR DIOXIDE AS BARIUM SULPHITE, THIOSULPHATE, AND SULPHUR.

OUTLINE OF TESTS.

As stated above the reaction between barium sulphide and sulphur dioxide may be expressed as follows:



Barium sulphide in water solution is more or less hydrolyzed so that the solution contains $\text{Ba}(\text{OH})_2 + \text{Ba}(\text{SH})_2$. Thus a reaction between BaS in solution and SO_2 solution may be written:



In this investigation it was necessary to determine the extent of these reactions under laboratory conditions and under the less completely controlled conditions likely to be met in large-scale operations. A knowledge of the physical characteristics of the precipitate formed under the different conditions was also essential.

Series of tests were made to determine the completeness of the reaction when the theoretical quantities were brought together (1) when both were in solution, (2) when the nearly pure solid barium sulphide was stirred into the sulphur dioxide solution, and (3) when the solid barium sulphide added was present in a reduced product of previous precipitations.

In other series similar precipitations were made when either barium sulphide or sulphur dioxide was present in excess.

In still other series repeated precipitations were made, the same mother liquor after filtration of the precipitate being used each time. The purpose of using the same mother liquor in the cycles was to determine the maximum concentration of barium salts that might be obtained in solution. These cyclic precipitations were first made on the small laboratory scale and later on the larger scale, when the barium compounds were used in cycles as well as the mother liquor.

The results of series of tests designated series A, B, C, and D, conducted on the small laboratory scale, are summarized below:

SERIES A PRECIPITATION TESTS.

In the Series A tests, standard solutions of barium sulphide were added at 20° C. to standard sulphur dioxide solutions in which there were no barium salts present.

The sulphur dioxide solutions were standardized against iodine and also against a standard alkali. The barium sulphide solutions were standardized against iodine and also by titration with a zinc hydroxide solution, nickel sulphate being used as an outside indicator.

Precipitations were made in glass-stoppered, wide-mouthed bottles, the solution being well shaken for several minutes during precipitation.

After the precipitate had been allowed to settle, an aliquot part of the solution was taken for the determination of excess uncombined sulphur dioxide. This was determined either by titrating with iodine both before and after the solution had been heated to 80° C., or by titrating with a standard alkali, methyl orange being used as an indicator. The precipitate was collected on a filter and washed once with hot water, and then dried at 100° C. Sulphur, barium sulphite, barium thiosulphate, and barium sulphate were then determined.

However, in some cases just the total acid soluble barium and insoluble barium were determined. Elemental sulphur was determined by leaching the precipitate with carbon bisulphide, evaporating off the carbon bisulphide on a tared watch glass, and weighing the residue sulphur.

As Ba(OH)_2 and Ba(SH)_2 are formed in dissolving BaS in water, it is necessary to keep the solution away from the atmosphere to prevent formation of the barium carbonate and loss of hydrogen sulphide. The sulphur dioxide solutions varied in strength from 0.66 gram to 37 grams of SO_2 per liter. The amount of barium sulphide added was between 37 and 118 per cent of that theoretically necessary for the reaction.

In these tests when the amount of barium sulphide theoretically required was added, the sulphur dioxide was completely neutralized, no uncombined sulphur dioxide or barium sulphide remaining in the solution. When the sulphide was added in an amount less than that theoretically required for the reaction, the excess uncombined sulphur dioxide in the solution was always less than the theoretical excess. This was due to the fact that the excess sulphur dioxide, to a certain extent, reacted with the precipitated barium sulphite,

or thiosulphate, dissolving these as a thionate or sulphite in which the atomic ratio of S to Ba was greater than 2:1. The higher the concentration of the sulphur dioxide in the solution before precipitation, the greater is the concentration of these barium salts in solution, as is indicated by the results of tests B-7, E-2, C-1, B-2, E-3, C-5, and C-9 given in Table 4 following. Also the greater the amount of sulphur dioxide in excess over that required for the reaction with barium sulphide, the greater is the concentration of the barium salts and the greater is the ratio of the sulphur to the barium in the solutions.

When an excess of barium sulphide was added, the iodine titration of the solution after precipitation was equivalent to more barium sulphide than the calculated excess. This excess iodine value was due to the fact that the solution contained a greater proportion of the SH ions than the OH ions. In several cases where there was only a slight excess of barium sulphide added, the addition of sulphur dioxide in an amount equivalent to the iodine value was sufficient to neutralize the solution. In the second precipitation, in which the solution was neutralized with sulphur dioxide, or a slight excess of sulphur dioxide was added, the precipitate contained more sulphur in proportion to the barium than did the first precipitate, which also indicated the presence of excess SH ions, after the first precipitation.

Irrespective of the concentrations of the sulphur dioxide or the barium sulphide solutions, the precipitates in all cases were principally the barium thiosulphate and sulphur ($2\text{BaS}_2\text{O}_3$ and S) with some barium sulphite and sulphur ($2\text{BaSO}_3 + 3\text{S}$). In some precipitates the sulphate was found, owing no doubt to oxidation during filtration and drying.

The thiosulphate and the sulphite were slightly soluble, the amount going into solution varying according to the original sulphur dioxide content of the solution, and the excess sulphur dioxide remaining in solution after reaction.

The precipitates were mostly flaky and flocculent, some of them containing a much smaller proportion of a crystalline precipitate. This crystalline precipitate was the sulphite.

When the sulphur dioxide solution was completely neutralized, or when the solution contained an excess of sulphur dioxide, the precipitate settled very rapidly. In a solution that was alkaline owing to an excess of sulphide, the precipitate settled very slowly, having colloidal characteristics.

TABLE 4.—Results of Series A precipitation tests.^a

BARIUM SULPHIDE IN SOLUTION ADDED TO SULPHUR DIOXIDE SOLUTION.

SULPHUR DIOXIDE SOLUTION, 36.8 GRAMS PER LITER; BARIUM SULPHIDE SOLUTION, 50.0 GRAMS PER LITER.

Test No.	Total SO ₂ used.	Total BaS used.	Proportion of BaS referred to percentage theoretically required.	SO ₂ per liter in solution after precipitation.		BaS per liter in solution after precipitation.		Ba per liter in solution.	Approximate atomic ratio of Ba in solution.	Per cent of total Ba in solution.	Per cent of total Ba in precipitate.
				Theoretical.	Determined.	Theoretical.	BaS equivalent to I ₂ titration.				
	Grams.	Grams.	P. ct.	Grams.	Grams.	Grams.	Grams.	Grams.			
B-4.....	3.68	7.68	118	0.00	0.00	5.23		5.26	2.5:1	20	80
B-3.....	3.68	6.84	105	.00	.00	1.71	2.85	4.63	2.2:1	18	82
B-7.....	3.68	6.48	100	.00	.00	.00	.00	2.99	2.0:1	12	88
B-8.....	3.35	5.90	100	.00	.00	.00	.00	2.64	2.0:1	11	89
B-2.....	3.68	5.90	91	1.65	.65	.00	.00	4.03	2.2:1	16	84
B-5.....	3.68	5.20	80	3.90	1.68	.00	.00	5.95	2.3:1	27	73
B-1.....	3.68	4.76	73	5.36	3.78	.00	.00	7.17	2.3:1	33	67
B-8.....	3.68	3.56	55	10.3	8.93	.00	.00	8.96	2.5:1	50	50

SULPHUR DIOXIDE SOLUTION, 18.0 GRAMS PER LITER; BARIUM SULPHIDE SOLUTION, 31.7 GRAMS PER LITER.

E-1.....	1.80	3.49	110	0.00	0.00	1.52	2.10	3.90	2.3:1	29	71
E-2.....	1.80	3.17	100	.00	.00	.00	.00	2.10	2.0:1	16	84
E-3.....	1.80	2.85	90	.95	.90	.00	.00	2.95	2.2:1	24	76
E-4.....	1.80	2.54	80	2.00	1.55	.00	.00	3.75	2.3:1	34	66
E-5.....	1.80	2.22	70	3.18	2.28	.00	.00	5.10	2.3:1	48	52

SULPHUR DIOXIDE SOLUTION, 4.4 GRAMS PER LITER; BARIUM SULPHIDE SOLUTION, 12.3 GRAMS PER LITER.

C-4.....	0.88	1.72	112	0.00	0.00	0.53	0.90	0.96	2.2:1	15	85
C-1.....	.88	1.54	100	.00	.06	.00	.00	1.40	2.0:1	34	66
C-5.....	.88	1.35	88	.35	.31	.00	.00	1.82	2.3:1	48	52
C-2.....	.88	1.23	80	.57	.56	.00	.00	1.84	2.3:1	55	45
C-6.....	.88	.96	62	1.19	1.12	.00	.00	1.75	2.3:1	59	41
C-3.....	.88	.80	52	1.58	1.43	.00	.00	1.50	2.4:1	58	42
C-7.....	.88	.57	37	2.91	2.20	.00	.00	1.20	2.4:1	98	2

SULPHUR DIOXIDE SOLUTION, 0.7 TO 1.5 GRAMS PER LITER; BARIUM SULPHIDE SOLUTION, 12.3 GRAMS PER LITER.

C-8.....	0.15	0.21	80	0.45	0.42	0.00	0.00	0.46	2.0:1	32	68
C-11.....	.15	.185	70	.70	.59	.00	.00	.65	2.0:1	50	50
C-9.....	.066	.104	90	.062	.060	.00	.00	.28	2.0:1	36	64
C-10.....	.070	.099	80	.130	.100	.00	.00	.24	2.1:1	32	68
C-12.....	.070	.088	70	.197	.145	.00	.00	.32	2.1:1	49	51

^a Tests by A. E. Wells, W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwardes.

SERIES B PRECIPITATION TESTS.

In the Series B tests, powdered barium sulphide was added to the sulphur dioxide solution which contained no barium salts. When powdered barium sulphide was used for precipitation, the precipitate always was mostly the thiosulphate and sulphur ($2 \text{BaS}_2\text{O}_3 + \text{S}$);

but the SO_2 consumption per unit of BaS added was variable, depending to a great extent on the degree of fineness to which the barium sulphide had been pulverized, and on the rapidity with which the sulphide was stirred into the solution. Very often the free sulphur dioxide would be completely removed from solution before all the sulphide had dissolved, even when only 80 per cent of the theoretically required amount of the sulphide had been added, and later as this sulphide dissolved, the solution became alkaline.

In Table 5 following are given the data obtained in tests when barium sulphide (95 per cent BaS and 5 per cent $\text{BaSO}_4 + \text{BaSO}_3$) pulverized to 200 mesh was used as the precipitant. The solid material was incorporated into the solution by vigorous shaking in a bottle.

TABLE 5.—Results of Series B precipitation tests.^a

[Solid barium sulphide (95 per cent BaS, 200 mesh) added to sulphur dioxide solution, which before precipitation contained no barium salts.]

Test No.	SO_2 in solution per liter.	BaS added, per cent of theoretical required.	Excess SO_2 per liter.		Ba per liter in solution after precipitation.	Atomic ratio of S Ba in solution.
			Calculated.	Determined.		
	<i>Grams.</i>		<i>Grams.</i>	<i>Grams.</i>	<i>Grams.</i>	
1.....	36.8	100	^b 0.0	^b 0.0	4.25	1.6:1
2.....	36.8	90	3.7	2.5	5.10	2.5:1
3.....	36.8	80	7.4	5.1	6.85	3.2:1
4.....	36.8	70	11.1	7.8	9.15	4.5:1
5.....	18.0	100	^b 0.0	^b 0.0	3.56	1.8:1
6.....	18.0	90	1.8	.0	3.95	3.1:1
7.....	18.0	80	3.6	1.9	4.50	3.2:1
8.....	18.0	70	5.4	2.8	5.80	3.5:1
9.....	7.3	100	^b 0.0	^b 0.0	1.26	1.8:1
10.....	7.3	88	.88	.30	1.62	2.2:1
11.....	7.3	43	4.16	2.20	4.10	3.5:1
12.....	4.4	100	^b 0.0	^b 0.0	1.15	1.9:1
13.....	4.4	90	.44	.00	1.50	2.2:1
14.....	4.4	80	.88	.30	1.60	2.3:1
15.....	.7	100	^b 0.0	^b 0.00	.60	2.0:1
16.....	.7	80	.14	.05	.95	2.0:1

^a Tests by A. E. Wells, W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwardes.

^b Solution contained excess BaS.

Table 6 following gives a brief summary of the results obtained from two series of precipitations made by shaking in a bottle reduced barium sulphate containing 70 per cent barium sulphide with sulphur dioxide solution. In the first series the material was pulverized to 100 mesh, and in the second was pulverized only to 60 mesh.

The other 30 per cent of this reduced material was barium oxide, barium carbonate, barium sulphate, and coke ash. Special tests showed that both the oxide and the carbonate reacted very slowly with the dilute sulphur dioxide solution and hence could not account for the rapid elimination of sulphur dioxide from the solution.

TABLE 6.—*Results of precipitation tests^a in which barium sulphide (70 per cent BaS, 100 or 60 mesh) was added to sulphur dioxide solution containing no barium salts.*

Test No.	SO ₂ per liter in solution at start.	BaS added, per cent of theoretical required.	SO ₂ per liter in solution at end.		Ba per liter in solution.	Atomic ratio of S to Ba in solution.	Remarks.
			Calculated.	Determined.			
	<i>Grams.</i>		<i>Grams.</i>	<i>Grams.</i>	<i>Grams.</i>		
1-0.....	9.0	100	0.00	0.00	1.52	1.9 : 1	Material used 70 per cent. BaS, ground to 100 mesh.
1-1.....	9.0	93	.63	.00	1.56	2.2 : 1	
1-2.....	9.0	83	1.53	.00	2.19	2.65 : 1	
1-3.....	9.0	72	2.51	.00	2.91	3.12 : 1	
1-4.....	9.0	62	3.42	.60	3.69	3.02 : 1	
1-5.....	9.0	52	4.32	1.52	4.06	3.10 : 1	
2-1.....	10.1	100	.00	b 0.00	(c)	(c)	Material used 70 per cent BaS, ground to 60 mesh.
2-2.....	10.1	96	.40	b 0.00	(c)	(c)	
2-3.....	10.1	85	1.50	.00	2.98	2.3 : 1	
2-4.....	10.1	65	3.53	.00	4.80	3.0 : 1	
2-5.....	10.1	53	4.75	.15	6.11	3.1 : 1	

^a Tests by A. E. Wells, W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwards.

^b Solution contained excess BaS.
^c Not determined.

SERIES C PRECIPITATION TESTS.

In the Series C tests, powdered barium sulphide was added to sulphur dioxide solution which contained barium salts before precipitation. The results of these tests can be summarized briefly by stating that when the sulphur dioxide solution contained originally some barium salts, the tendency for the free sulphur dioxide in the solution to react immediately with the first products of the reaction and to dissolve them was decidedly less than when the barium sulphide was added to a "straight" sulphur dioxide solution containing no barium salts.

The tendency for the removal of the sulphur dioxide from the solution by a reaction with the first precipitation products was less as the barium concentration in the solution was greater.

With a barium concentration of about 10 grams per liter, and the ratio of sulphur to barium in solution being 3 to 4 atoms of S to 1 atom of Ba, the reaction between barium sulphide and sulphur dioxide took place nearly according to the theoretical reaction, requiring 2 molecules of BaS to 3 molecules of SO₂.

Unless there was considerable excess sulphur dioxide in the solution, the barium sulphide added reacted only with the theoretical amount of sulphur dioxide to form thiosulphate and sulphur (2BaS₂O₃ + S). However, with considerable excess sulphur dioxide after the sulphide had dissolved, or with strong sulphur dioxide solution before precipitation, there was a tendency for the resolution of some of the precipitated barium salts and the formation of soluble thionates or sulphites.

SERIES D PRECIPITATION TESTS.

In the Series D tests, repeated precipitations were made, the same solutions being used throughout a test. This series was in fact only a continuation of Series C, carried out with a sufficient number of repeated precipitations to indicate to what extent the barium salts would accumulate in the solution.

The precipitations were made from solutions of about 2 liters in volume.

In one set of tests, practically the theoretically required amount of barium sulphide was added each time. The amount of barium in solution increased as the number of precipitations until a maximum content of about 15 grams of barium per liter of solution was obtained. The ratio of barium to sulphur in solution at this point was approximately four atoms of sulphur to one atom of barium. The specific gravity of the solution was 1.045. This solution was allowed to stand for several days, when there was a crystallization of some of the barium salts, principally the thiosulphate.

Other series of cycles were run in which at certain precipitations insufficient barium sulphide was added, and at other precipitations the sulphide was added in excess. In these series, the barium content of the solution after precipitation was greater or less according to whether considerable excess sulphur dioxide was in the solution after precipitation, and according to the rapidity with which the material was stirred into the solutions.

In these laboratory cycle series the barium content of the solution never went above 18 grams per liter, and the ratio of sulphur to barium never went above 5 atoms of S to 1 atom of Ba.

The specific gravity of these solutions never exceeded 1.05. Table 7 following shows the results of two tests, which are given as examples of the data obtained from several tests.

TABLE 7.—Results of Series D precipitation tests.^a

[Repeated precipitations.]

TEST 1.

Precipitate No.	SO ₂ per liter in solution at start.	BaS added, per cent of theoretical required.	SO ₂ per liter in solution at end.		Ba per liter in solution.	Ratio of S to Ba in solution.	Specific gravity of solution.
			Calculated.	Determined.			
	Grams.		Grams.	Grams.	Grams.		
1.....	9.2	100	0	(b)	2.25	1.6:1	1.010
2.....	10.0	100	0	(b)	3.90	2.2:1	(c)
3.....	10.0	100	0	(b)	5.88	3.0:1	(c)
4.....	9.3	100	0	0.0	6.76	3.4:1	1.020
5.....	9.0	100	0	.2	8.10	4.0:1	(c)
6.....	10.1	100	0	.5	9.45	4.5:1	(c)
7.....	9.5	100	0	.2	12.20	(c)	1.040
8.....	11.5	100	0	.0	13.40	5.1:1	(c)
9.....	9.5	100	0	.2	15.50	(c)	1.040
10.....	11.5	100	0	.0	14.20	4.7:1	(c)
11.....	10.2	100	0	.2	14.80	(c)	(c)
12.....	9.5	100	0	.1	15.25	(c)	1.045

TEST 2.

1.....	9.6	50	4.8	2.3	5.50	3.1:1	1.015
2.....	10.2	100	.0	(b)	4.86	3.0:1	(c)
3.....	9.5	80	1.9	.0	8.22	3.5:1	(c)
4.....	11.5	85	.6	.0	9.06	3.8:1	(c)
5.....	9.5	80	1.9	.8	8.10	4.1:1	1.020
6.....	11.4	118	.0	(b)	7.15	4.0:1	(c)
7.....	9.0	140	.0	(b)	6.90	4.0:1	(c)
8.....	11.2	60	4.8	3.2	10.10	4.5:1	(c)
9.....	11.2	100	.0	.0	9.15	4.5:1	(c)
10.....	11.2	100	.0	.0	9.85	4.5:1	1.030

^a Tests by A. E. Wells, W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwardes.

^b Excess BaS.
^c Not determined.

SUMMARY OF RESULTS OF TESTS IN SERIES A TO D.

The small-scale laboratory tests embraced in series A to D indicated that the main reaction between barium sulphide and sulphur dioxide was $2\text{BaS} + 3\text{SO}_2 = 2\text{BaS}_2\text{O}_3 + \text{S}$. The thiosulphate and sulphur were the main part of the precipitate, whether the sulphide was added in solution or in finely powdered form.

When solution was added to solution in nearly equivalent amounts there was only a slight tendency for complex soluble thionates, or sulphites, to form, although a small percentage of the thiosulphate dissolved in the mother liquor.

When solid barium sulphide was added to the sulphur dioxide solution, unless the solution already contained between 10 and 15 grams of barium per liter, the SO_2 equivalent per unit of BaS was greater than the theoretical, this excess sulphur dioxide consumption being due to the sulphur dioxide reacting with the barium sulphite, or to the thiosulphate being precipitated to form complex soluble thionates or soluble sulphites. This tendency, however, would not be a serious factor in practical work, as the concentration of the barium salts remained comparatively low and the maximum specific gravity of the solution was about 1.05.

In the tests it was noted that the settling of the precipitate was most rapid when the solution was nearly neutral after precipitation and when the specific gravity of the solution was about 1.03.

LARGE-SCALE PRECIPITATIONS.

The large-scale precipitations were made with the apparatus shown in Plate I, A (p. 10).

Sulphur dioxide from the furnace gases was absorbed in the tower by the mother liquor, which was pumped to the top of the tower and ran out at the bottom into the tubs, which held 70 liters (18.5 gallons) of solution. The sulphur dioxide content of the solution was determined immediately after a tub had been filled, and then a weighed amount of a product from the reduction process, containing barium sulphide, was incorporated into the solution by vigorous stirring and agitation. This stirring was done by hand with a paddle for a period of 5 to 10 minutes. The precipitate, in most cases practically all $2\text{BaS}_2\text{O}_3 + \text{S}$, was allowed to settle, and the mother liquor was then siphoned off by a pump and lifted to the top of the tower for the absorption of more sulphur dioxide.

The precipitate was filtered and dried, and the sulphur was distilled from it. After distillation of the sulphur, the residue ($\text{BaSO}_3 + \text{BaSO}_4$) was mixed with pulverized coke and reduced in a furnace to the sulphide. Thus, in these tests not only was the mother liquor used in cycles but the barium compounds were used repeatedly.

In the first cycles, the sulphur dioxide consumption factor per unit of reduced material was a variable, as in the small laboratory-scale precipitation. After several cycles of the mother liquor, when the barium content of the solution had been built up to 8 to 10 grams per liter, the sulphur dioxide consumption factor of the reduced material became more constant. The addition of an insufficient amount of the reduced material to take care of the sulphur dioxide according to the reaction $2\text{BaS} + 3\text{SO}_2$, resulted simply in the presence of excess sulphur dioxide after precipitation. The barium content of the mother liquor was about 8 grams per liter when the solution was nearly neutral after precipitation, but was as high as 12 grams per liter when considerable excess sulphur dioxide was present.

When the products of reduction of previous precipitates were used, especially after the barium compounds had been carried through several cycles, it was rather difficult to calculate from analysis the exact sulphur dioxide equivalent per unit of reduced product. The factor had to be determined by trial, an approximate calculated factor being used at the start. This was due to the fact that most reduction products contained barium oxide and barium carbonate, especially after the first few cycles, when the reductions were effected in a direct-fired furnace. Some of the oxide was readily soluble in water, and this reacted readily in the sulphur dioxide solution, forming the sulphite. However, a considerable part of the oxide was slowly soluble. In some products a small proportion of the sulphide was nearly insoluble.

The slowly soluble compounds in the reduced product reacted slowly with the sulphur dioxide solutions, especially with the dilute solution remaining after the greater part of the sulphur dioxide had been removed by the readily soluble barium sulphide. Most of the nearly insoluble barium oxide was so insoluble that the first part to go into the solution frequently reacted with the excess sulphur dioxide to form a soluble bisulphite, removing the free sulphur dioxide before the rest of the barium oxide dissolved.

However, when trial had shown the sulphur dioxide equivalent of the reduced product under the conditions for precipitation, there was no difficulty experienced in causing the sulphur dioxide to be completely precipitated from the solution and in keeping the barium content of the solution down to less than 10 grams of barium per liter. For example, in one series, the mother liquor was used throughout 58 cycles. At the end the solution contained 7.6 grams of barium per liter, the ratio of sulphur to barium being equal to 5.8:1 and the specific gravity of the solution being 1.032.

The ratio of sulphur to barium in the mother liquor at times was as high as 7 atoms of sulphur to 1 atom of barium. Some attempts were made to determine the exact composition of the barium salts in

solution by causing crystallization of these salts. However, they were unstable, and every attempt to evaporate the solution by heating resulted in the precipitation of sulphur and barium sulphate, and in some tests in the evolution of sulphur dioxide.

Rhombohedral crystals were obtained by cooling the solution to a temperature slightly below that of precipitation. These crystals contained 2 atoms of sulphur to 1 atom of barium, corresponding to the thiosulphate BaS_2O_6 , or to the barium dithionate BaS_2O_8 .

In these large-scale precipitations, it was noted that considerable heat was liberated in the reactions between the solid barium sulphide and the sulphur dioxide. Thus, for example, 70 liters of solution containing 9.8 grams of SO_2 per liter, or a total of 0.680 kilogram SO_2 , was precipitated with 1.7 kilograms of reduced material containing 70 per cent (1.20 kilograms) of BaS . The rise in temperature of the solution was 2.5°C . ($30.5^\circ\text{--}28^\circ\text{C}$). The approximate heat liberated was 175 kilogram calories, or 103 kilogram calories per kilogram of reduced material.

PRACTICAL CONSIDERATIONS REGARDING PRECIPITATION.

In practice the precipitant will have to be incorporated mechanically with the solution as the latter runs into the agitating tank. This can be done, as planned by the Thiopen Co., by pumping a part of the solution containing the sulphur dioxide through an incorporator with an adjustable feed, and then mixing the emulsion and the rest of the solution through a centrifugal pump discharging into an agitator. A skilled attendant will be required to regulate the feed of precipitant to meet the sulphur dioxide content of the solution. The precipitant will probably have to be used in large batches of fairly constant composition and sulphur dioxide combining value, so that the regulation of feed can be adjusted without much difficulty.

The heat generated in the precipitation is of some practical importance, as this heat must be removed either in the settling tank, or in the storage tanks; otherwise the solution will become so warm that its absorbent efficiency will be greatly decreased.

SETTLING PRECIPITATE AND DECANTING MOTHER LIQUOR.

For practical use the data obtained on the settling of the precipitate in the large-scale precipitations in the tubs are much more valuable than those obtained in the small-scale laboratory experiments. However, the small-scale experiments indicated that settling took place most rapidly from a nearly neutral solution, and was slowest from a solution containing excess barium sulphide, that is, an alkaline solution.

In the large-scale work the precipitations were made about as follows: Five tubs of 70-liter capacity each were used for precipitation

and settling. Each tub was filled in 7 to 10 minutes with the mother liquor containing sulphur dioxide running from the towers. The analyses for sulphur dioxide in the solution and the weighing of the reduced material for precipitation took about two minutes. The solid precipitate was then stirred vigorously into the solution for 6 to 8 minutes until the next tub following was nearly filled. Generally, this tub was again stirred for several minutes until the second following tub was nearly filled and was then allowed to settle. By the time the mother liquor of this tub had to be drawn for return to the towers, about 15 or 20 minutes later, the precipitate had almost completely settled, leaving a supernatant liquor in which there were some flakes of sulphur floating.

Thus, in these tests, a settling capacity equivalent to the volume of solution coming from the towers in 20 minutes was ample to allow satisfactory settling.

When the precipitation was carried out properly, that is, when the solid precipitant was stirred vigorously into the solution and the solution after precipitation was nearly neutral, about 90 per cent of the precipitant settled 12 inches to the bottom of the tub within three minutes. This settling was even more rapid after a precipitation in which a considerable part of the solid material added did not react with the sulphur dioxide in the solution, that is, was comparatively insoluble.

The supernatant liquor after settling 10 minutes still contained some light, flaky precipitate, largely sulphur, in most cases less than 1 per cent of the total. This could be handled without any difficulty through the gear pumps, centrifugal pump, or plunger pump used at different times. This light precipitate did not settle in the absorption tower, nor did the amount in the mother liquor increase markedly, unless, as stated above, the solution became alkaline.

The mother liquor could be drained off to the point where the residual precipitated sludge contained not over 50 per cent water.

FILTRATION OR DEWATERING OF PRECIPITATE.

In most of the large-scale tests the precipitated sludge was washed or shoveled from the precipitation or settling tubs onto a canvas filter of about 0.279 square meter (3 square feet) area, under which there was maintained a vacuum of about 38 centimeters (18 inches) of mercury. Without a vacuum a cake about 1 centimeter ($\frac{1}{2}$ inch) thick was obtained in 10 minutes. With vacuum a 4-centimeter ($1\frac{1}{2}$ -inch) cake could be dewatered in 2 minutes, only about 25 per cent water being left in the cake. With a 38-centimeter vacuum, a 5-centimeter (2-inch) cake could be obtained easily.

It was found that the precipitate when once settled should be separated from the supernatant liquor and filtered within a few

hours. In several tests the precipitated sludge was allowed to stand 24 to 48 hours in contact with a neutral mother liquor solution. As a result the precipitate became colloidal and slimy and offered serious obstacles to filtration. One particular lot of precipitated sludge had stood in contact with the mother liquor for 24 hours. After decantation it contained 65 per cent water and 35 per cent solids. An attempt to filter this through a Merrill dry discharge press, with a filtering area of 9 square feet and under an air pressure of 30 to 35 pounds, gave only a $\frac{3}{4}$ -inch cake in $1\frac{1}{2}$ hours, the resultant cake containing 45 per cent water.

However, when the sludge was filtered soon after it had settled no difficulties whatever were experienced in the operation. The water content of the cake was reduced to as low as 18 per cent in several tests when air was drawn through the cake for 4 to 5 minutes after the greater part of the water had been removed. An average cake contained 20 to 26 per cent moisture.

In the dewatering as in the filtering operations, the rapidity was greater in the case of precipitated sludge containing other barium compounds than the precipitate; that is, the barium compounds that did not react with the sulphur dioxide.

The conclusions from all the work on filtering the precipitate were that if filtration was accomplished soon after settling no difficulty should be experienced in reducing the moisture to about 25 per cent. Standard dewatering apparatus, for example, an Oliver filter or a Merrill or a Kelly press, would be satisfactory.

DRYING PRECIPITATE.

A drying plate having an area of about 7.5 square meters (25 square feet) was built of 1-centimeter ($\frac{3}{8}$ -inch) iron plates. These plates were placed about 2.5 centimeters (5 inches) above the floor and gas burned beneath. The sides of this drying oven were of brick. The precipitate was stirred frequently and evaporation was assisted by blowing air over the surface. It was necessary to prevent the material from getting too hot locally, for in that event the sulphur started to distill and burn. It was noted that there was a slight distillation of sulphur even at 110°C .

In the product from this dry plate 6 to 25 per cent of the total barium was present as sulphite, 40 to 85 per cent as thiosulphate, and 7 to 34 per cent as sulphate. The product contained 6 to 10 per cent elemental sulphur. It was not necessary to regrind the dried precipitate before distillation.

In practice, drying would undoubtedly be best accomplished by steam coils, or by a special type of muffle furnace in which there would be no danger of local overheating, with attendant loss of sulphur.

DISTILLATION AND RECOVERY OF SULPHUR.

In the tests it was determined that by heating the dried precipitate in a retort to 450° to 460° C. all the elemental sulphur as well as one-half the sulphur in the barium thiosulphate could be distilled.

Only in the small laboratory experiments was the sulphur distilled and condensed under such conditions that the sulphur was completely recovered and a quantitative sulphur balance obtained.

In the laboratory tests distillations were effected in glass retorts placed in an electrically heated furnace and the sulphur was recovered in glass condensers. In most cases there was a slow stream of nitrogen or sulphur dioxide passed through the retort into the condenser to remove the sulphur vapors.

In the distillation tests the sulphur began to distill rapidly at 180° C., and from that temperature up to 450° C., the boiling point of sulphur, the rate of distillation depended on the temperature and the rapidity with which the vapors were removed from the retort.

In all cases, whether the precipitate was almost entirely ($2\text{BaS}_2\text{O}_3 + \text{S}$) or whether considerable ($2\text{BaSO}_3 + 3\text{S}$) was also present, the sulphur was rapidly and completely distilled at 450° C. The condensed sulphur was practically pure.

The residual product from the distillation was barium sulphite, sulphate, and in some cases a small amount of sulphide. Even when special care was taken to exclude air from the retort there was some slight oxidation of the sulphite to sulphate. Sometimes, when air was not completely removed or when air could get in contact with the hot residue after removal of the sulphur vapors, considerable sulphate formed.

Also, when the material was heated to a temperature higher than 450° C. there was some conversion of the sulphite to sulphide and sulphate according to the equation: $4\text{BaSO}_3 = \text{BaS} + 3\text{BaSO}_4$. In one product from a distillation at 600° C., 6 per cent of the barium was present as sulphide.

In the large-scale experiments, different methods for the distillation and condensation of the sulphur vapors were tried in a rough way, but no apparatus was built that gave complete recovery of the sulphur. Flowers of sulphur, or rubber-like modifications were obtained according as the vapors were allowed to condense slowly, or cooled quickly by precipitation on chilled surfaces. A discussion of the various methods used is not considered necessary as giving any data worth recording.

In order to effect a rapid distillation of the sulphur, the mass had to be stirred or agitated. Besides allowing the vapors to escape readily, the agitation prevented the mass from balling or caking.

Most distillations were condensed in an iron kettle placed in a brick furnace fired by gas. The temperature was measured in the mass

being distilled. By allowing sulphur vapors to burn a short distance above the material under treatment, the material was kept out of contact with the air. Other distillations were made in a 6-inch iron pipe, which was rapidly revolved in a brick furnace. In these distillations the sulphur vapors were condensed in an iron cylinder. The residual products were barium sulphite and sulphate, no thio-sulphate being present. In the products of the cyclic operations there was also the oxide and the carbonate that had not reacted with the sulphur dioxide.

Some of the products contained as much as 50 per cent of the barium as sulphate; others only about 10 per cent. In practical operations it would probably be safe to figure that 30 to 40 per cent of the barium in the product would be present as sulphate, and the rest as sulphite, except 2 or 3 per cent as sulphide.

The product was pulverized, and was readily mixed with pulverized coke for the subsequent reduction.

The tests indicated that in practice the distillation of the sulphur from the dried precipitate consisting of either $(2\text{BaS}_2\text{O}_3 + \text{S})$ or $(2\text{BaSO}_3 + 3\text{S})$ should give no difficulty. Necessarily, the furnace for distillation must be heated externally and be provided with apparatus for stirring the charge. For quickly removing the sulphur vapors from the distillation furnace, it would be advisable to keep a current of sulphur dioxide gas circulating through the distillation furnace into the condensation chamber.

REDUCTION.

Reduction of the barium sulphite and the barium sulphate was recognized at the beginning of the study of the Thiopen process, as being probably the most important factor in the whole process.

In answer to several inquiries concerning what was being accomplished at the present time in the reduction of barium sulphate at various plants where barium salts were being manufactured from barium sulphate, the information was gained that although 80 or 85 per cent of the barium was readily reduced to acid-soluble compounds, yet current practices rarely reduced more than 60 or 70 per cent of the barium to water-soluble barium sulphide.

Information gained from the literature ^a showed that it was difficult to reduce more than 80 per cent of the barium to the sulphide, and that a 70 per cent reduction was more often the best result obtained in practice.

In order that the barium shall be efficient as a precipitant during the Thiopen process cycles, the maximum reduction of the sulphite

^a Frazier, Schuyler, Calcining plant for barium sulphate: *Min. Ind.*, vol. 19, pp. 71-73; Fay, A. N., Roasting barytes: *Min. Ind.*, vol. 19, pp. 73-75; Toch, Maximilian, The barium industry in the United States since the European War: *Met. and Chem. Eng.*, vol. 14, January, 1916, pp. 47-49.

or the sulphate to the water-soluble sulphide must be attained in each cycle. Thus, it was necessary to determine the conditions under which the barium sulphite or sulphate was reduced most completely to the sulphide, and considerable experimental work has been conducted on that problem. As barium sulphate must be used at the start in the initial reduction, and was present to a large extent in all precipitates after distillation, and also as this material could be obtained in quantities much more readily than the sulphite, the greater part of the investigations dealt with the reduction of the sulphate.

Before beginning the discussion of the several series of reduction tests, it is advisable to discuss briefly the character of the products obtained, and the manner in which these products are classified in this discussion. The barium compounds found in the reduced products were as follows:

1. Compounds readily soluble in hot water—
BaS, BaS_x, and BaO.
2. Compounds difficultly soluble in hot water but readily soluble in dilute hydrochloric acid—
BaO, BaCO₃, BaSiO₃, BaS, and BaSO₃.
3. Compounds insoluble in acid—
BaSO₄.

DETERMINATION OF WATER-SOLUBLE BARIUM.

The reduced product, ground to 100-mesh, was leached in a flask with 250 parts of water, which was just below the boiling point and filtered into a flask which was tightly stoppered while the solution was cooling. An aliquot part of the solution was then titrated with iodine, and in another aliquot part, barium and sulphur were determined. As a check for the iodine value, that is, the barium sulphide content, the material was frequently treated directly in a flask with excess iodine for about five minutes, and then the excess iodine was determined. In many cases excess barium hydroxide was present in the water solution. In other products there was a slight excess of sulphur over that required for the barium to form sulphide, indicating a polysulphide. In several products of reductions at high temperatures, 1050° to 1200° C., and in which excess carbon was present, the presence of the carbide was indicated, but was not determined quantitatively.

BARIUM COMPOUNDS SLOWLY SOLUBLE OR INSOLUBLE IN HOT WATER, BUT READILY SOLUBLE IN DILUTE ACID.

In many products from the first reduction, a large part of the oxide present was slowly soluble in hot water, but was readily soluble in dilute hydrochloric acid.

In most of the products of the reductions that were effected in a direct-fired furnace at high temperatures, and in which slight sintering had taken place, a small amount of the sulphide was found in the water-insoluble part. In some products 4 to 6 per cent of the total sulphide was found to be slowly soluble in water. The solubility of this material was increased somewhat by finer grinding. The insolubility of this part of the sulphide was due probably to its being covered with a film of the oxide. In a typical average product from the first reduction of barium sulphate 97 per cent of the sulphide and 40 per cent of the oxide was readily soluble in hot water, the rest being found in the acid-soluble part.

In this water-insoluble but acid-soluble part was also found the carbonate in varying amounts. The products from slow reductions at low temperatures contained more carbonate than did the products from rapid reductions at high temperatures.

Few products contained more than a trace of the sulphite or the thionates.

These insoluble barium products reacted very slowly with the sulphur dioxide in the dilute solutions. Thus, in certain cycles described later the proportion of these compounds increased to the point that over 40 per cent of the total barium was present in the form of soluble or insoluble oxide and insoluble carbonate. For example, as an extreme case, in the reduced product from the fourth cycle of one series of cyclic operations the barium compounds were found to be soluble to the extent shown below. These barium compounds were first pulverized to 100 mesh and then leached in a flask with 250 parts of hot water, subsequent leaching being made in a beaker.

Results of leaching tests of barium compounds.^a

Material.	Barium compound.	Barium recovered, percentage of total barium content in original material.
A. Soluble products in solution obtained by leaching pulverized barium compounds with 250 parts of hot water for 15 minutes.	BaS BaO	22.3 24.9
B. Soluble products obtained by leaching residue from A with 250 parts of hot water for 2 hours.	BaS + BaO	4.2
C. Soluble products obtained by leaching residue from B with boiling water for 10 hours.	BaS + BaO	8.0
D. Soluble products obtained by leaching residue from C with boiling water for 12 hours.	BaS + BaO	6.2
Total soluble products from solutions B, C, and D (24 hours' leaching).	BaO	3.9
E. Soluble products obtained by leaching residue from D with dilute HCl.	BaO, BaCO ₃ , and BaSiO ₃	14.5
F. Insoluble residue in dilute HCl solution (E).	BaSO ₄	8.6 9.8

^a Tests by R. Buhman and V. P. Edwards.

RESULTS OF REDUCTION TESTS.

The reduction tests were divided into five series, designated series 1 to 5, and the results are here considered under each series heading.

SERIES 1 TESTS.

In the Series 1 tests were embraced tests to determine the relative rates of reduction of barium sulphate by hydrogen, carbon monoxide, and a mixture of these gases with other hydrocarbons at different temperatures. In the tests precipitated barium sulphate (99+ per cent BaSO_4) was placed in a quartz tube of about 12 mm. (0.47 inch) diameter, and heated in an electrically heated tube furnace. The barium sulphate was placed between two asbestos plugs, filling the tube for a distance of about 30 mm. (1.18 inches), but was sufficiently loose to allow the passage of gases through the material. The gases were passed through the tube at the rate of about 2 c. c. per minute. The temperature range was between 600° and $1,050^\circ$ C. The data obtained probably did not represent the chemical equilibrium at any temperature. The comparative rate of reaction at different temperatures was obtained, however, as were data concerning the character of the gaseous products.

REDUCTION WITH HYDROGEN.

When hydrogen was used as the reducing agent there was practically no reaction at any temperature below 550° C. Around 600° C. the reaction was extremely slow, less than 5 per cent of the hydrogen that passed through the plug reacting. The gaseous product contained considerable hydrogen sulphide. At 650° C. about 15 per cent of the hydrogen reacted. At 700° C. about 35 per cent reacted. Between 600° and 700° C. the amount of hydrogen sulphide formed was a maximum. At 800° C. practically all the hydrogen had reacted and little hydrogen sulphide was present in the gaseous product. In general, these data checked those of a previous investigation.^a

In the products from these reductions with hydrogen the ratio of sulphur to barium was less than one atom of sulphur to one atom of barium.

If the reductions were effected slowly at lower temperatures, that is, between 600° and 750° C., the loss of sulphur was greater than if the reductions were effected more rapidly at a higher temperature, as between 900° and $1,000^\circ$ C. In one test lasting for an hour at $1,000^\circ$ C. 94 per cent of the sulphate was reduced to sulphide and 6 per cent to oxide. This was the maximum reduction obtained with the use of hydrogen.

^a Marino, I. L., [Technical preparation of barite]: *Am. Chem. Soc. Abs.*, vol. 7, 1913, p. 3202; [Reduction of sulphate of alkaline earths with various gases]: *Gazz. Chim. Ital.*, vol. 43, pp. 416-22

In some correlated tests it was found that between 700° and 900° C. water vapor reacts slowly with barium sulphide, forming sulphur dioxide, hydrogen sulphide, and barium oxide. For example, through a reduced product of about 20-mesh size, containing 89 per cent of the barium as sulphide, 7 per cent as oxide and carbonate, and 4 per cent as sulphate, was passed at 900° C. a mixture of about 16 per cent water vapor and 84 per cent nitrogen. About 1 per cent of the water vapor reacted, forming nearly two volumes of sulphur dioxide per volume of hydrogen sulphide. The reaction between water vapor and barium sulphide at other temperatures was not determined.

REDUCTION WITH CARBON MONOXIDE.

With carbon monoxide the reduction was slow below 650° C. and was fairly rapid at 750° C., about 70 per cent of the carbon monoxide reacting. With temperatures up to 900° C., however, there was present in the gaseous product some unconsumed carbon monoxide. Above that temperature the gaseous product was almost entirely carbon dioxide. In these reduced products the ratio of sulphur to barium was slightly less than 1 to 1, showing that some sulphur had been removed in the gaseous products.

In this series, as in tests with hydrogen as the reducing gas, the loss of sulphur was less when reduction was effected rapidly at higher temperatures, that is, above 900° C., than when effected slowly at lower temperatures. Thus the products of reduction effected rapidly at high temperatures contained the smaller percentages of water-insoluble barium oxide.

REDUCTIONS WITH CITY GAS.

In a series of 20 reductions an excess of city gas of approximately the following composition was passed at the rate of 2 c. c. per minute through the charge in the quartz tube:

Composition of city gas used in reduction experiments.

	Per cent.
Methane (CH ₄).....	6.4
Ethane (C ₂ H ₆).....	30.4
Hydrogen (H).....	41.4
Carbon monoxide (CO).....	13.4
Carbon dioxide (CO ₂).....	4.0
Nitrogen (N).....	4.2
Oxygen (O).....	.2

The results of the experiments are given in Table 8 following:

TABLE 8.—*Results of tests involving reduction of precipitated BaSO₄ by city gas in quartz tube, externally heated.^a*

Temperature.	Time.	Per cent of total Ba in product present as—		
		Water-soluble BaS and BaO.	Water-insoluble, acid-soluble BaO and BaCO ₃ .	Acid-insoluble BaSO ₄ .
°C.	Minutes.			
650- 700	60	2.5	4.0	93.5
	240	58.0	15.8	26.2
	330	72.5	18.2	9.3
700- 750	60	8.5	6.8	84.7
	240	75.0	13.8	11.2
750- 800	60	20.6	8.4	71.0
	240	88.2	11.0	0.8
800- 850	60	50.8	7.8	41.4
	240	91.5	8.3	0.2
850- 900	60	55.5	6.8	37.7
	240	91.8	8.0	0.2
900- 950	60	66.2	6.2	28.6
	120	85.0	7.3	7.7
	240	92.5	7.5	0.0
950-1,000	60	74.7	4.8	20.5
	120	91.5	6.5	2.0
	240	93.2	6.8	0.0
1,000-1,050	60	85.2	5.0	9.8
	120	92.0	6.5	1.5
	240	93.0	7.0	0.0

^a Tests by W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwardes.

The results presented in the preceding table indicate that in the tests in which city gas was used the percentage of barium in the product present in water-insoluble, acid-soluble compounds was less when reductions were effected in a short period of time at temperatures above 900° C. than if effected over a longer period of time at lower temperatures. For example, the products obtained when the material was treated for 330 minutes at 650° to 700° C. should be compared with the products obtained in a reduction at 1,000° to 1,050° C., lasting only 60 minutes.

It is also evident that any tendency for a reaction between the reducing gases and the sulphide, resulting in the formation of water-insoluble barium compounds, is slight. For example, the proportion of these compounds in the products of reductions at 900° to 1,050° C. for 60, 120, and 240 minutes should be compared.

SERIES 2 TESTS.

In the Series 2 tests barium sulphate was reduced by carbon. Finely pulverized carbon, either coke or charcoal, was intimately mixed with the barium sulphate and heated for varying periods of time at different temperatures in crucibles or tubes placed in muffle furnaces.

The theoretical amount of carbon required for the reduction is 10.3 per cent of the weight of BaSO₄ if the carbon is completely oxidized

to the dioxide, or is 20.6 per cent, if the carbon is oxidized only to the monoxide. In the tests the carbon added varied between 10 and 40 per cent of the BaSO_4 .

The temperature range was between 700° and $1,250^\circ \text{C}$. The time allowed for reduction varied between 15 and 360 minutes.

In all but a few of the tests, in which the time allowed for reduction was an hour or more, the mixtures, weighing 50 to 100 grams, were placed in 10-gram or 20-gram fire-clay crucibles and heated in a muffle furnace. The muffle had been heated to the required temperature before the crucibles were placed in it, and thus the crucibles and their contents were heated quickly. When the time allowed for reduction was less than one hour it was rather difficult to get satisfactorily concordant data with this size of charge and method of heating. In order to reduce the heating period to a minimum, these reductions were made with much smaller charges, 5 to 10 grams, and were effected in porcelain crucibles.

In the data following most of the temperatures recorded are the temperatures measured near the crucibles during the period of reduction. Several temperatures, however, were measured in the charge itself, after the charge had come to the temperature of the muffle. In the tests, care was taken to have the heating of the muffle so regulated that the temperature did not vary more than 15° or 20°C . either way from that desired. Thus, in the tables following, if the temperature is given as 700°C ., the temperature was in reality between 680° and 720°C ., which was sufficiently close for the purposes of the investigations.

In making these reductions, it was very important as nearly as possible to keep the air from coming into contact with the material. Thus, the crucibles were covered with a clay cover ground to fit the crucibles. In some tests a charcoal cover about 2 mm. (0.04 inch) thick was placed on the charge. Even with the tight covering, in some of the tests there was a slight reversion of the sulphide back to the sulphate, owing possibly to a slight porosity of the crucible walls allowing air to pass through. Some of the data concerning this reversion are given later.

TABULATED RESULTS OF TESTS.

In Tables 9 and 10 following are summarized the results of 76 tests made under conditions that were closely similar, so that the results are comparable. In these tests, the carbon added was 20.6 per cent of the weight of the barium sulphate. The barium sulphate used was the precipitated material. The carbon was pulverized to pass a 160-mesh screen, and was mixed with sulphate to the degree that, to the eye, the mixture looked homogeneous.

The variables were the time allowed for reduction and the temperature. Many of the figures given in the tables are the averages of several determinations that in most cases gave concordant results.

TABLE 9.—*Results of reduction tests of barium sulphate showing total barium in reduced product in form of water-soluble barium sulphide.^a*

Time allowed for reduction.	Per cent of total barium present in reduced product as water-soluble barium sulphide at temperature of—							
	700° C.	750° C.	800° C.	850° C.	900° C.	950° C.	1000° to 1100° C.	1100° to 1200° C.
<i>Minutes.</i>								
15					48		75	96
30	12	29	51	58	65	76	84	97
60	26	35	61	69	85	89	96	
90	28	55	63	78	87	92		
120	41	59	68	85	88	95		
240	65	78	82	86				
360	77							

^a Tests by W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwardes.

TABLE 10.—*Results of reduction tests of barium sulphate showing total barium in reduced product in compounds soluble in dilute hydrochloric acid.^a*

Time allowed for reduction.	Per cent of total barium present in reduced product as compounds soluble in dilute hydrochloric acid at temperature of—							
	700° C.	750° C.	800° C.	850° C.	900° C.	950° C.	1000° to 1100° C.	1050° to 1150° C.
<i>Minutes.</i>								
15					54		80	98
30	21	38	57	65	72	80	89	100
60	37	47	70	78	94	94	100	
90	40	69	75	87	97	97		
120	54	73	85	94	99	100		
240	80	93	98	98				
360	93							

^a Tests by W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwardes.

It is to be noted that there is less difference between the percentage of the barium present in acid-soluble compounds and that present as water-soluble barium sulphide in the products of reduction effected at higher temperatures for a short period of time, than in products from reductions at lower temperatures over longer periods.

Most of the products from the 76 reduction tests summarized above contained excess carbon. Other series of tests were made to determine to what extent a decrease in the amount of carbon present in the charge affected the reduction.

When the tests were made with a porcelain crucible fitted with an ordinary porcelain cover, there was frequently sufficient air leaking into the crucible to allow either burning of a small percentage of the carbon, or reoxidization of a small amount of the reduced sulphide

which thus became sulphate. As the data from these tests were, therefore, not entirely accurate, they are not presented here.

Similar reductions were repeated in porcelain crucibles, when a light layer of carbon was placed on top of the charge and other precautions were taken to keep the air from coming into contact with the charge during cooling, etc. Although it was determined that the carbon cover did have some slight reducing action on the charge, yet it was decided that the data obtained were sufficiently accurate for recording in this report, and they are given in Table 4 following.

Later reduction tests, effected in an atmosphere of nitrogen, which was neither oxidizing or reducing, gave results that checked the above data fairly closely.

TABLE 11.—*Results of tests involving reduction of barium sulphate at different temperatures in porcelain crucibles with variable amounts of carbon.*^a

Temperature.	Time.	Parts of carbon per 100 parts of barium sulphate.	Per cent of total Ba in product, present in—			Per cent of total carbon consumed.	Parts of carbon consumed to 100 parts of BaSO ₄ reduced to BaS.
			BaS.	Acid-soluble compounds.	Insoluble compounds.		
* C.	Minutes.						
800	60	10.5	50	58	42	69	14.5
800	60	21.0	51	56	44	49	20.2
800	120	10.5	62	70	30	83	14.2
800	120	21.0	66	65	32	65	20.5
850	60	10.5	66	70	30	96	13.2
850	60	15.5	60	65	35	65	16.8
850	60	20.5	60	70	30	58	19.8
850	120	10.3	78	85	15	97	12.9
850	120	15.5	76	82	18	94	17.0
850	120	20.6	74	80	20	70	19.4
900	60	10.3	80	85	15	100	12.9
900	60	15.5	81	83	17	87	16.7
900	60	20.6	76	80	20	72	19.4
900	120	10.3	80	86	14	100	12.9
900	120	15.5	84	90	10	99	16.5
900	120	20.6	83	87	13	80	19.8
950	30	10.3	80	83	17	100	12.5
950	30	20.6	84	87	13	78	19.0
950	60	40.0	78	79	21	59	12.8
950	60	10.3	94	97	3	100	20.2
950	60	20.6	95	96	4	93	23.4
950	60	30.0	82	82	18	61	11.2
1,050	30	10.3	82	86	14	100	18.1
1,050	30	15.5	96	98	2	100	18.7
1,050	30	20.6	95	96	4	89	11.5
1,150	20	10.3	90	90	10	100	15.6
1,150	20	15.5	99	99	1	90	18.2
1,150	20	20.6	99	100	0		

^a Tests by W. Freeman, R. Buhman, C. E. Brandt, and V. P. Edwards.

^b This factor may be slightly low, owing to a slight reducing action of the carbon cover.

In another set of reduction tests the barium sulphate was mixed intimately with finely pulverized carbon and placed in a porcelain tube in an electrically heated tube furnace. The tube was filled with nitrogen gas at the start, and the gases evolved were discharged through a water seal or into caustic-potash solutions. Thus neither oxygen nor any reducing gases other than the carbon monoxide from

the carbon came in contact with the barium sulphate. The results are given in Table 12 following:

TABLE 12.—*Results of tests involving reduction of barium sulphide by carbon in tube furnace, surrounded by atmosphere of nitrogen.^a*

Temperature.	Parts of carbon per 100 parts of BaSO ₄ in charge.	Per cent of total carbon consumed.	Per cent of BaSO ₄ reduced to BaS.	Carbon consumed per 100 parts of BaSO ₄ reduced.
* C.				
800	10	80	58	13.8
800	10	100	70	14.3
800	20	63	64	19.8
900	10	95	70	12.5
900	10	100	74	13.5
950	10	97	75	12.9
950	10	100	75	13.3
950	15	90	82	16.4
950	14	100	96	14.7
950	15	100	90	16.7
1,000	10	100	80	12.5
1,000	15	100	96	15.6
1,000	20	84	95	17.6

^a Tests by W. Freeman and V. P. Edwardes.

COMMENTS ON RESULTS OF TESTS.

The tabulated data indicate that the carbon efficiency was greatest when the charge contained the smallest percentage of carbon. In general, somewhat higher carbon efficiency was obtained in the reductions effected at the higher temperatures than at the lower.

Although the highest carbon efficiency (about 12 to 12.5 per cent) was obtained when the carbon present was about 10 per cent of the barium sulphate, the theoretical amount if the carbon was oxidized completely to carbon dioxide, yet with this amount of carbon the maximum reduction of the barium sulphate at temperatures up to 1,000° C. was only about 80 per cent. With 15 per cent carbon present in the charge, the reduction was nearly complete when the carbon was consumed.

When the carbon was present to the extent of 20 per cent of the BaSO₄, the carbon efficiency was much lower irrespective of whether the reduction was at high or low temperatures, but slightly higher at the high temperatures than at the low.

From these data it is to be noted, also, that with an excess of carbon the reduction seemed to proceed slightly slower than did the reduction with the lesser amount of carbon present. This is shown especially in the results of series at 850°, 900°, and 950° C., shown in Table 4.

However, it must be recognized that the fineness to which the material is pulverized, and the intimacy with which the sulphate and carbon are mixed, are factors influencing these results, as is the depth of material through which the carbon monoxide gas must pass

before leaving the charge. For example, in several tests "straight" barium sulphate was placed on top of the charge containing 20 per cent carbon. This barium sulphate cover was largely reduced by the carbon monoxide coming up from the charge proper, thus increasing the efficiency of the carbon when credited with the whole amount of barium reduced.

Summarized briefly, the data from the Series 2 tests show that when the carbon and the barium sulphate were finely pulverized and intimately mixed, and the reduction was effected in a muffle or crucible type of furnace, in the absence of outside air or of products of combustion of fuel, and at temperatures between 850° and $1,000^{\circ}$ C., a high reduction (90 per cent or better) could be obtained when the carbon added was about 15 per cent of the barium sulphate. With the addition of less carbon, although the carbon efficiency was greater, the reduction was not so complete. With the use of more than 15 per cent carbon, the carbon efficiency decreased, unless provision was made to utilize the carbon monoxide issuing from the charge by passing it through another charge of barium sulphate.

Different kinds of barium sulphate gave different results in the reductions carried on under the same conditions. For example, the following materials, all finer than 200-mesh, were reduced at the same time under the same conditions, that is, a temperature of 900° C. and a reduction period of 120 minutes, with the following results:

Results of reduction tests of different kinds of barium sulphate.

Material.	Per cent of total barium in product in -		
	Water-soluble compounds.	Water-insoluble, acid-soluble compounds.	Acid-insoluble compounds.
Precipitated BaSO_4 , 99.6+ per cent.....	88	10	2
Pulverized barytes, 98 per cent BaSO_4	78	13	9
Pulverized barytes, 92.5 per cent BaSO_4	72	18	10
Precipitated $\text{BaSO}_3 + \text{BaSO}_4$	93	7	0

In the pulverized barytes containing 98 per cent barium sulphate the impurities were barium carbonate, iron, and silica. In the barytes containing only 92.5 per cent barium sulphate there was about 3 per cent iron and 3 per cent silica.

Some comparative tests were made to determine the effect of iron on the reduction of barium sulphate. Four charges were made up in which iron to the extent of about 6 per cent was mixed with the precipitated barium sulphate. In one charge the iron was added as ferric sulphate, in another as ferrous ammonium sulphate, in a third

as ferrous sulphate, and in a fourth as ferric oxide. Two charges of the pure precipitated barium sulphate were reduced under the same conditions—a temperature of 850°C . and a reduction period of 120 minutes. The average results of four tests were as follows:

Results of reduction tests in which iron was added to reduction charge.

Material reduced.	Per cent of total barium present in—		
	Water-soluble compounds.	Water-insoluble, acid-soluble compounds.	Insoluble compounds.
Pure BaSO_4	80	10	10
BaSO_4 with 6 per cent of iron present.....	68	17	15

REOXIDATION OF BARIUM SULPHIDE TO SULPHATE.

It has been stated that barium sulphide, when at red heat, is readily reoxidized to the sulphate. The data from several of the crucible-reduction tests were found to be of no value because inadequate precautions were taken to prevent the oxygen from getting into the crucible. An example of the effect of small air leakages into the crucible is given below. These results were obtained when pulverized barytes (98 per cent BaSO_4) was reduced with 20 per cent willow charcoal at $1,000^{\circ}\text{C}$.

Results of reduction tests in which air leaked into reduction crucible.

Time of reduction.	Per cent of barium in product present in—			Condition of cover.
	Water-soluble BaS.	Water-insoluble, acid-soluble compounds.	Insoluble BaSO_4 .	
<i>Minutes.</i>				
60	94	6		Tight.
60	78	3	21	Loose.
120	56	4	40	Loose.
120	35	3	62	Very loose.

The reoxidation product is the sulphate, not the oxide. This fact is shown in the results presented in the foregoing table, also in the results of some special tests that were made to determine the product of oxidation of barium sulphide. In these tests material containing 95 per cent of the barium as the sulphide was heated in the air for varying periods of time at different temperatures. In most cases practically all the sulphide was reconverted to the sulphate, and only small amounts of oxide were formed. The maximum

conversion to oxide was about 7 per cent, and was obtained when the material was heated to 800° C. for two hours.

SERIES 3 TESTS.

In the Series 3 tests, barium sulphate mixed with carbon and made into briquets was reduced in a shaft furnace heated by oil. Precipitated barium sulphate was mixed with lampblack carbon from the local gas works, and the mixture was pressed into the form of cupels, 2.5 cm. (1 inch) in diameter and 3.2 cm. (1¼ inches) high. These cupels were dried and then were charged into a small shaft furnace 30.5 cm. (12 inches) in diameter, and 76.2 cm. (30 inches) high. This furnace was heated to about 950° to 1100° C. by oil, the products of combustion passing through the shaft and maintaining strongly reducing conditions around the briquets. The furnace was kept filled with briquets during a test, more briquets being added at the top as the reduced product was withdrawn below. The length of time the briquets were in the furnace varied between one and one-half and three hours. During an average run of six hours there was about 80 kilograms (176 pounds) of the briquets reduced. The best average product obtained from any of the runs contained 82.5 per cent of the barium as sulphide, 13.8 per cent as oxide and carbonate, and 3.7 per cent as insoluble sulphate. The best product from any charge during this run contained 87 per cent of the barium as sulphide. During this run the maximum furnace temperature was 1150° C. The average carbon monoxide content of the gases in the shaft of the furnace was 4 per cent.

In this shaft furnace several runs also were made with briquets of pulverized commercial barytes (92.5 per cent BaSO_4) and lampblack carbon. These briquets were 6.36 cm. (2½ inches) long, 6.35 cm. (2½ inches) in diameter, and weighed each about 370 grams (or 0.81 pound). With these briquets unsatisfactory reductions were obtained, even though the temperature of the furnace was raised to 1100° C., and broken pieces of coke were charged with the briquets. The best products contained only about 70 per cent of the barium as sulphide. Also, unless the large briquets were broken and chilled immediately on being removed from the furnace, they retain their heat for a long time, and the barium sulphide to a great extent oxidized back to the sulphate.

One charge of reduced briquets weighing about 25 kilograms, and containing at the time of withdrawal from the furnace about 70 per cent of the barium as sulphide, was allowed to stand unbroken in an iron container for about 15 hours. At the end of this time, the briquets in the center of the container were still glowing. The

outside cooled briquets contained less than 15 per cent of the barium as sulphide, the rest having oxidized back to sulphate.

SERIES 4 TESTS.

In the Series 4 tests barium sulphate was reduced in a multiple-hearth furnace.

DESCRIPTION OF MULTIPLE-HEARTH REDUCTION FURNACE.

The furnace (Pl. I, *B*, and Pl. II, *A*) was designed by Mr. Utley Wedge, and was loaned to the bureau's representatives for the investigations.

The furnace was 24 inches (61 centimeters) in inside diameter and 34 inches (86.3 centimeters) high; that is, between the bottom of the cast-iron drying hearth 7 (Pl. I, *B*) to the top of the bottom hearth 12. Beneath the bottom hearth was a calcine pit 12 inches deep. Above the drying hearth were placed the driving gears, roller bearings, etc. The total height over all was 6 feet (1.83 meters).

The walls of the furnace were of specially molded fire brick 8 inches (20.3 centimeters) thick, containing openings for gas burners, pyrometer tubes, gas-sampling tubes, and mica windows, or peepholes. Outside the brick walls of the furnace was placed a covering of magnesia board $1\frac{1}{2}$ inches thick. This magnesia was covered with a layer of asbestos cement one-eighth inch thick, over which was placed a thin layer of Portland cement.

In the center of the furnace was hung a cast-iron column, 1 (Pl. I, *B*), $3\frac{1}{2}$ inches (95.2 centimeters) in diameter, which was keyed into the center of a gear wheel, 36 inches (0.915 meter) in diameter, revolving on roller bearings, 3. These roller bearings were carried on a horizontal cast-iron bedplate, 4, resting on four short adjustable pins, 5. The pins were screwed into sockets in the flange 6 of the drying plate 7. This drying plate was bolted to four steel columns, 8, placed outside the walls of the furnace. The upper bearing for the center column was carried on a steel frame, 9, which was also bolted to the four steel columns. The large gear, meshed with a worm drive, 10, was bolted to the bedplate 4. At one end of this worm-drive shaft was placed a pulley, 11, over which belt connection was made to a driving motor.

To the revolving center column inside the furnace were attached three cast-iron hearths (13, II, *A*; see also figs. 2 and 3) 22 inches (55.9 centimeters) in diameter. Thus, there was an annular space 1 inch (2.54 centimeters) wide between these hearths and the furnace walls. Between these movable hearths and below the lowest were placed three stationary hearths 12 (Pl. II, *A*), resting on shelves

provided on the brick walls. The opening at the center of these hearths was 6 inches (15.25 centimeters) in diameter, leaving an annular space 1 inch wide between the stationary hearths and the

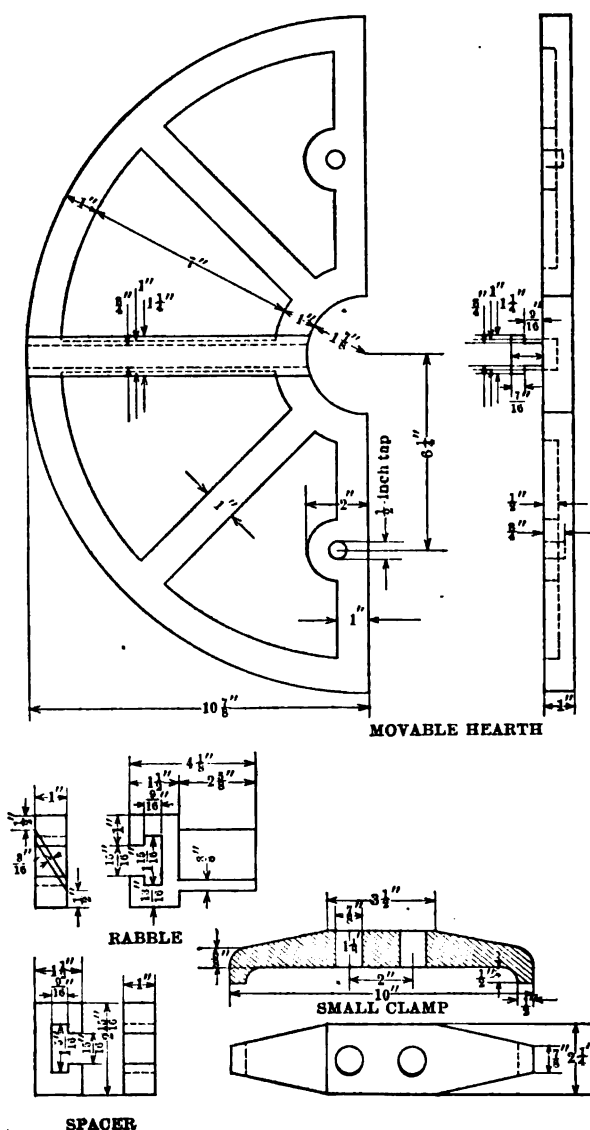


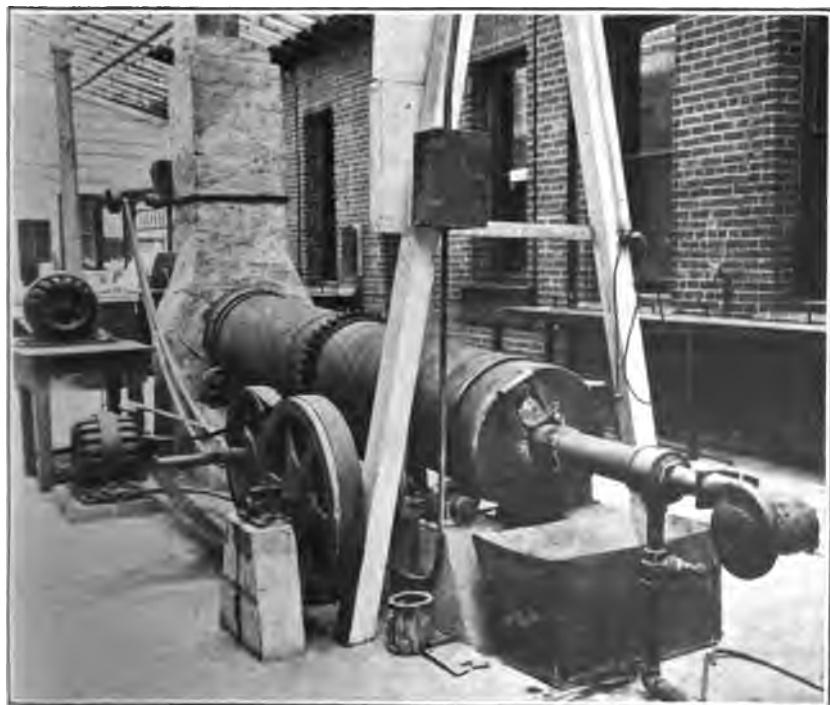
FIGURE 2.—Details of a movable hearth in multiple-hearth furnace.

center columns. The total effective hearth area was 16.5 square feet (1.53 square meters).

Rabbles (15, Pl. II, A; see also figs. 2 and 3) were attached to ribs (16, Pl. II, A) cast on the underside of the hearths. The rabbles



A. INTERIOR OF FURNACE, SHOWING CAST-IRON HEARTHS AND RABBLES.



B. CEMENT KILN USED IN REDUCTION TESTS AT HIGH TEMPERATURES.

stirred and moved the ore across the hearths immediately below. The ore on the cast-iron drying floor (7, Pl. I, B) was moved by the rabbles (17, Pl. I, B) revolving with the center column. From the drying floor the ore fell to the center of the top movable hearth and was moved by the rabbles suspended from the stationary drying

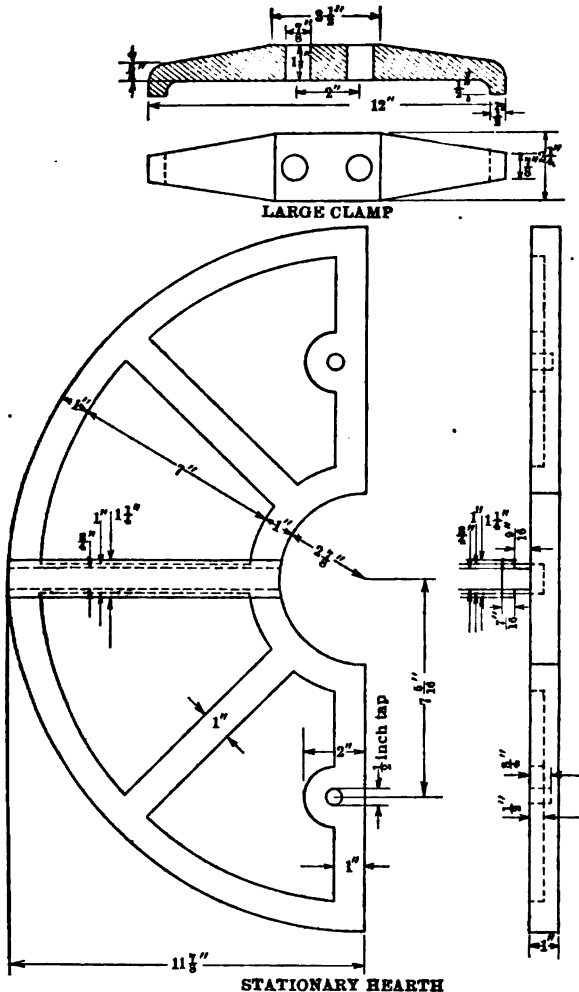


FIGURE 3.—Details of a stationary hearth in multiple-hearth furnace.

hearth to the outside of the top revolving hearth, falling onto the second hearth, a stationary hearth. It was then moved by rabbles attached to the top movable hearth to the center of the stationary hearth and fell onto the third hearth, a moving hearth. The ore was thus moved through the furnace in a manner similar to that in a regular large-size multiple-hearth roasting furnace.

The cast-iron hearths lasted for many tests when the temperatures were not over 800° C., but when temperatures above 900° C. were maintained they warped and cracked and had to be replaced after eight or nine 12-hour runs.

Bristol pyrometers (18, Pl. I, B) were inserted through the walls of the furnace for measuring the temperature on the first, third, and fifth hearths. Gas samples were also drawn from the furnace at these hearth levels. Peepholes (19, Pl. I, B) were provided at several points for observing the charge in the furnace.

Movable brick plugs (20, Pl. I, B) were provided in the walls of the furnace, which gave access to the interior for cleaning off the hearths. The movable plugs were somewhat necessary, as some charges had a tendency to ball up in front of the rabbles.

The furnace was heated by oil fuel or by city gas. In most of the tests gas was used. The gas was burned through Meker burners (21, Pl. I, B), with a 20-pound air pressure at the burners.

It is not considered that the fuel consumption in this furnace furnished data for calculating the fuel that would be required in a standard furnace of this type if properly designed to prevent radiation losses, etc. The center revolving column was water cooled. The cooling water removed one-fifth to one-third of the total heat supplied to the furnace by the combustion of the gas. In most of the reduction tests the furnace was operated under only a slight draft—just enough to allow the products of combustion, etc., to be removed from the furnace through the flue (22, Pl. I, B).

In straight roasting, when pyrites containing 35 to 40 per cent sulphur were being roasted, the furnace, once heated, could be run several hours, roasting at the rate of 30 or 35 pounds (13.6 to 16.1 kilograms) per hour without the use of extraneous fuel.

SUMMARY OF REDUCTION TESTS IN FURNACE.

Following are summarized briefly 20 tests made in the furnace. In making the reduction tests the variables were as follows: (1) Length of time the material was in the furnace; (2) the furnace temperature; (3) reducing conditions.

REDUCTION PERIOD.

The length of time the material was in the furnace was controlled by the rate at which the charge was fed and by the speed of revolution of the center column. The rate of feeding was varied between 5.6 to 19.8 pounds (2½ to 9 kilograms) per hour, and the length of time the material was in the furnace varied between 2 and 3½ hours.

FURNACE TEMPERATURES.

The variations in the furnace temperatures were as follows:

Furnace temperatures in reduction tests.

Part of furnace.	Minimum temperature.	Maximum temperature.	Average temperature.
First hearth.....	°C. 180	°C. 485	°C. 360
Third hearth.....	560	905	800
Fifth hearth.....	840	920	850

REDUCING CONDITIONS.

In all tests the quantity of city gas used was in excess of that required for the combustion to maintain the proper temperatures. In the 20 tests summarized, no free oxygen was allowed in the furnace. The carbon monoxide content of the gases taken from the furnace on the third and fifth hearth ranged between 0.5 and 5 per cent. The fuel-gas consumption was between 3 and 6 cubic feet per minute. The main consideration in these tests was to obtain the maximum reduction rather than the highest efficiency of the reducing agent—carbon or gas.

These 20 reduction tests may be divided into the following groups:

1. Tests involving reduction of precipitated barium sulphate (99 per cent BaSO_4) intimately mixed with lampblack carbon, heavy oil residuum, pulverized coke, or powdered charcoal.

2. Tests involving reduction of precipitated barium sulphate (99 per cent BaSO_4) with excess hydrogen, carbon monoxide, or hydrocarbons entering the furnace through the burners, no carbon being mixed with the material.

3. Tests involving reduction of commercial pulverized barytes (98 per cent BaSO_4 , as used in paints by the W. P. Fuller Co.) with coke, lampblack, carbon, or like material mixed intimately with the barytes.

TABULATED RESULTS OF TESTS.

The results of the tests are given in Table 13 following:

71284°—17—4

TABLE 13.—*Results of reduction tests in multiple-hearth furnaces.*

[Tests by A. E. Wells, A. L. Tuttle, W. Freeman, R. Buhman, and C. E. Braudt.]

REDUCTION OF PRECIPITATED BARIUM SULPHATE (99+ PER CENT BaSO_4) INTIMATELY MIXED WITH LAMPBLACK CARBON, HEAVY-OIL RESIDUUM, PULVERIZED COKE, OR POWDERED CHARCOAL.

Test No.	Parts of carbon per 100 parts of BaSO_4 in charge.	Character of carbon used.	Rate of charging mixture per hour.	Rate of gas consumption per minute.	Approximate length of time material was in furnace.	Total time of test.	Average temperature.			Maximum temperature in furnace.	Average per cent of CO in furnace gases (third hearth).	Analysis of product of test. ^a			
							Top hearth.	Third hearth.	Fifth hearth.			Total Ba water soluble.	Total Ba acid soluble.	Total Ba insoluble BaSO_4 .	Maximum per cent of Ba water soluble in hourly samples.
1	20	Lampblack.	Kilograms.	Cu. feet.	Hours.	Hours.	°C.	°C.	°C.	°C.		Per cent.	Per cent.	Per cent.	43
2	16	do.	8.9	5.2	2	2	190	590	810	840	4.5	18	35	66	73
3	20	do.	5.0	5.0	2	9	270	590	890	890	5.0	71	88	12	79
4	25	do.	3.5	6.0	2	12	300	640	890	890	4.7	75	93	7	67
5	25	Pulverized coke.	3.5	3.7	3	9	270	570	890	895	1.6	55	78	22	78
6	17	do.	3.5	4.6	3	10	420	760	895	905	2.0	77	92	8	82
7	18	do.	2.8	3.5	3	10	370	810	890	905	.5	80	98	2	80
8	15	do.	3.5	3.5	3	12	370	790	895	910	2.0	76	94	6	80
							460	890	890	900		77	90	10	

REDUCTION OF PRECIPITATED BARIUM SULPHATE (99+ PER CENT BaSO_4) BY EXCESS H_2CO OR HYDROCARBONS ENTERING FURNACE THROUGH BURNERS. NO CARBON MIXED WITH MATERIAL.

Test No.	Parts of carbon per 100 parts of BaSO_4 in charge.	Character of carbon used.	Rate of charging mixture per hour.	Rate of gas consumption per minute.	Approximate length of time material was in furnace.	Total time of test.	Average temperature.			Maximum temperature in furnace.	Average per cent of CO in furnace gases (third hearth).	Analysis of product of test. ^a			
							Top hearth.	Third hearth.	Fifth hearth.			Total Ba water soluble.	Total Ba acid soluble.	Total Ba insoluble BaSO_4 .	Maximum per cent of Ba water soluble in hourly samples.
B-3	20	Lampblack.	Kilograms.	Cu. feet.	Hours.	Hours.	°C.	°C.	°C.	°C.		Per cent.	Per cent.	Per cent.	43
B-4	16	do.	8.9	5.2	2	2	190	590	810	840	4.5	18	35	66	73
B-5	20	do.	5.0	5.0	2	9	270	590	890	890	5.0	71	88	12	79
B-6	25	do.	3.5	6.0	2	12	300	640	890	890	4.7	75	93	7	67
B-7	25	Pulverized coke.	3.5	3.7	3	9	270	570	890	895	1.6	55	78	22	78
		do.	3.5	4.6	3	10	420	760	895	905	2.0	77	92	8	82
		do.	2.8	3.5	3	10	370	810	890	905	.5	80	98	2	80
		do.	3.5	3.5	3	12	370	790	895	910	2.0	76	94	6	80
							460	890	890	900		77	90	10	

REDUCTION OF PULVERIZED COMMERCIAL BARYTES (AS USED IN PAINTS BY W. P. FULLER CO.), WITH COKE, LAMPBLACK CARBON, ETC., MIXED INTIMATELY WITH BARYTES.^c

C-1	30	Pulverised coke.....	3.5	3.8	28	124	370	760	870	910	0.5	51	59	41	60
C-2	20	do.....	3.5	4.5	28	8	410	790	895	900	1.0	55	75	35	63
C-3	20	Lampblack.....	3.5	5.0	3	6	380	790	890	900	2.5	68	80	20	72
C-4	20	Pulverised coke.....	3.0	4.2	3	7	410	790	895	900	1.0	68	80	20	72
D-1	0.0		3.0	5.0		10	480	850	890	910	.8	53	64	36	70
D-2	0.0		3.0	5.0		11	410	820	890	900	2.0	67	75	25	72
D-3	0.0		3.0	5.5		10	420	810	900	905	4.2	68	82	18	

^a Note maximum reduction.^b Products from all tests except 6 and 8 contained excess carbon.^c In tests designated by a figure preceded by the letter D, no carbon was used, reduction being effected by gases alone.

COMMENTS ON RESULTS OF TESTS.

The results presented in Table 13 indicate that with precipitated barium sulphate the highest reductions were obtained when the material was mixed with carbon either as pulverized charcoal or lampblack, and when the higher temperatures were maintained in the furnace. Under these conditions the percentage of excess reducing gases present in the furnace could be decreased to a low figure.

The pulverized commercial barytes were more difficult to reduce than was the precipitated barium sulphate.

When the maximum temperature in the furnace was above 850°C., the material was slightly sintered into balls about 0.3 centimeter (1.8 inches) in diameter, like cement cinders. The slightly sintered balls formed in the first reductions were somewhat broken and the material was pulverized readily for subsequent use as a precipitant in the solutions.

It is to be regretted that reduction tests at higher temperatures—above 900°C.—could not have been made with this type of furnace. However, refractory hearths to withstand higher temperatures over longer periods of time were not available.

SERIES 5 TESTS.

In the Series 5 tests barium sulphate was reduced in a cement kiln.

Through the courtesy of the chemistry department of the University of California, a small cement kiln (Pl. II, *B*) was made available for further work on the reduction of barium sulphate at high temperatures.

This cement kiln was 3.66 meters (12 feet) long, 48.3 centimeters (19 inches) in outside diameter, and 25.4 centimeters (10 inches) in inside diameter. The number of revolutions was 1.15 per minute, and the pitch was 1:18.

In the tests with this furnace, city gas or oil fuel was used for heating. The lower end of the kiln was closed by a sheet-iron plate covered with asbestos. Through this plate a gas burner was inserted, the gas being blown in with air under low pressure. A small gate was provided in this sheet-iron plate to allow drawing out the reduced product intermittently.

The gaseous products of the kiln were discharged into a brick stack. The draft at the furnace was reduced to a minimum, so that the reducing conditions in the furnace could be controlled by the air and gas entering through the burner pipe. Even with the draft reduced to a minimum, there was considerable barium sulphate carried out of the furnace by the gases. In one test when oil was used and was being discharged through the burner by a high air pressure, directly along the center line of the kiln, the loss of barium sulphate was high—about 30 per cent. When this oil-and-air mixture was discharged at right angles with the center line, striking against the sides of the kiln, the loss was greatly reduced, but was not less than 5 per cent under any conditions.

Provision was made for taking samples of gas at several points along the length of the kiln, and also for measuring the temperatures at these points by thermocouples.

The charge to the furnace was fed in by hand at the back of the furnace.

In these reduction tests, the temperature at the hottest part of the furnace was as high as $1,100^{\circ}\text{C}$., and averaged $1,050^{\circ}\text{C}$. For over half of the furnace the temperature was above 850°C . The average time the charge was in the furnace was $2\frac{1}{2}$ hours. The average rate of feeding the charge to the furnace was 15 kilograms (33 pounds) per hour, ranging between 10 and 20 kilograms per hour. The best reductions were obtained when the furnace was kept well filled with the charge.

In these tests, the material used was finely pulverized barytes (98+ per cent BaSO_4 , being material used in paints by the W. P. Fuller Co.), as large quantities of the precipitated barium sulphate could not be obtained. In six tests city gas was used, and in one test oil was used for fuel. The main object in the tests was to obtain a high reduction of the barium to the water-soluble sulphide, the obtaining of a higher efficiency of the reducing agent being a secondary consideration. However, one test (5-7), was conducted for the purpose of obtaining a higher fuel efficiency than was obtained in the other tests. This test showed that almost as high a reduction could be obtained with a much less fuel consumption.

None of these tests furnished conclusive data for calculating the amount of fuel that would be required for effective reduction in a commercial sized kiln.

The results of the tests with the cement kiln are given in Table 14 following:

TABLE 14.—Results of tests involving reduction of commercial pulverized barytes in cement kiln.

[Tests by A. E. Wells, W. Freeman, R. Buhman, and V. P. Edwardes.]

Test No.	Parts carbon per 100 parts BaSO_4 in charge.	Character of carbon used.	Rate of charging mixture per hour.	Rate of gas consumption per minute.	Average temperature of exit gases.	Average temperature at middle of kiln.	Approximate maximum temperature in kiln.	Average per cent of CO in furnace gases at middle of furnace.	Total time of test.	Analysis of product of test.			
										Total Ba water soluble.	Total Ba acid soluble.	Total Ba in soluble BaSO_4 .	
			Kilos.	Cu. ft.	$^{\circ}\text{C}$.	$^{\circ}\text{C}$.	$^{\circ}\text{C}$.		Hours.	P. ct.	P. ct.	P. ct.	
5-1	20	Pulverized coke.....	10.0	5.4	450	850	1,050	3.8	22 $\frac{1}{2}$	89	98	98	2
5-2	15	do.....	10.0	5.6	460	880	1,050	4.2	10	87	97	97	3
5-3	20	Pulverized charcoal.....	15.0	6.1	420	880	1,100	3.1	5	90	98	98	3
5-4	16	do.....	15.2	6.2	400	860	1,050	4.2	5	89	98	98	3
5-5			15.5	6.2	510	900	1,100+	6.2	4	89	78	78	22
5-6	18	Pulverized coke.....	16.0	(b)		1,360	(c)		2	84	96	96	4
5-7	13	Coke.....	18.7	3.8	370	840	1,050	1.5	4	82	90	90	10

* Product clinkered and unreduced BaSO_4 was left in the interior of the clinker.

b Rate of oil consumption, 3 gallons per hour.

c No data.

SUMMARY OF DATA CONCERNING REDUCTION OF BARIUM SULPHATE IN
FIVE SERIES OF TESTS.

Summarizing briefly the data obtained in the five series of reduction tests outlined above, it may be stated that the maximum reduction of the barium sulphate to sulphide was obtained at the higher temperatures—that is, around 1000°C ., and when the reductions were effected in an indirect-fired furnace, as, for example, a muffle furnace. When reductions were effected in this type of furnace, 15 or 16 per cent carbon gave the highest fuel efficiency consistent with the completeness of the reduction of the barium to the sulphide.

In reductions effected in a direct-fired furnace, as a cement kiln, a multiple-hearth roasting furnace or a shaft furnace in which hydrogen, hydrocarbons, or the products of combustion of the fuel—water and carbon dioxide—were brought into contact with the sulphate or sulphide, there was formed a larger proportion of barium compounds, insoluble in water, than was formed in a furnace indirectly fired. Thus, although the barium compounds in the best products from a direct-fired furnace were 90 to 95 per cent soluble in acid, yet the barium present as the water-soluble sulphide was not more than 85 to 87 per cent of the total.

By effecting the reductions rapidly at high temperatures—that is, above 1000°C .—the proportion of these water-insoluble barium oxides and carbonates was less than that formed in reductions at lower temperatures over a longer period of time.

Below 750°C . the reduction by carbon or reducing gases was too slow to be considered commercially feasible.

From these data it is concluded that in order to effect the maximum reduction of the sulphate to the water-soluble sulphide the reductions should take place in an indirect-fired furnace at high temperatures. This means a muffle-type furnace or a furnace built along the lines of a modern coal-gas producer.

TESTS TO DETERMINE DECREASE OF BARIUM SULPHIDE IN
REDUCED PRODUCTS FROM CYCLIC OPERATIONS.

As stated in the above discussion concerning reduction, in none of the reductions effected in the shaft furnace, the multiple-hearth furnace, or the cement kiln where these reducing gases or water vapor came in contact with the material did the final reduced product contain less than 8 per cent of the barium as a water-insoluble product, which was principally the oxide.

In the regular precipitations of the cyclic operations, these slowly soluble or insoluble barium compounds reacted rather slowly with the sulphur dioxide in the dilute solutions left after the greater part of the sulphur dioxide had reacted with the readily soluble sulphide.

Thus in the cycles a large proportion of these compounds was not converted into the sulphite or sulphate in the precipitation operation, but was carried around in the cycles as "inactive" material. As in each reduction stage there was a certain amount of these "inactive" compounds formed, the proportion of these gradually increased to such a point that, in several cyclic tests, more than half of the material used was of little or no service as a precipitant.

Summaries of these data from three cyclic tests are given in Tables 15, 16, and 17, following:

CYCLIC OPERATIONS IN WHICH REDUCTIONS WERE MADE IN THE
MULTIPLE-HEARTH FURNACE.

In Table 15 are presented results showing the decrease in the proportion of barium sulphide with a corresponding increase in the proportion of the oxide and carbonate in the products of reduction during four cycles in reduction tests effected in the multiple-hearth furnace.

TABLE 15.—*Results of cyclic operations in which reductions were made in multiple-hearth furnace.^a*

Material reduced.	Per cent of total Ba present as—				Reduced product No.
	BaS readily soluble in H ₂ O.	BaO readily soluble in H ₂ O.	BaO, BaS, BaCO ₃ , etc., insoluble in H ₂ O, soluble in HCl.	Insoluble BaSO ₄ .	
Pure BaSO ₄	74	2	12	<i>Per cent.</i> 12	1
Precipitate formed by adding reduced product 1 to SO ₂ solution.....	70	5	18	7	2
Precipitate formed by adding reduced product 2 to SO ₂ solution.....	66	6	24	4	3
Precipitate formed by adding reduced product 3 to SO ₂ solution.....	57	8	32	3	4

^a Tests by A. E. Wells, A. L. Tuttle, W. Freeman, R. Buhman, and C. E. Brandt.

CYCLE OPERATIONS IN WHICH REDUCTIONS WERE MADE IN CEMENT
KILN.

In Table 16 are given summarized data concerning the products obtained during the operation of the process through six complete cycles, the apparatus shown in Plate I, *A*, being used for absorption, precipitation, decantation, filtration, and drying. The distillations were made in an iron kettle, and reductions were effected in the cement kiln shown in Plate II, *B*.

The reduced product used as a precipitant in the first cycle contained 84.1 per cent of the total barium as sulphide, 5.1 per cent as water-soluble oxide, 8.7 per cent in water-insoluble but acid-soluble compounds, and 2.1 per cent as sulphate. The ratio of sulphur to barium in this material was 0.88 atom of sulphur to 1 atom of barium.

stirred and moved the ore across the hearths immediately below. The ore on the cast-iron drying floor (7, Pl. I, B) was moved by the rabbles (17, Pl. I, B) revolving with the center column. From the drying floor the ore fell to the center of the top movable hearth and was moved by the rabbles suspended from the stationary drying

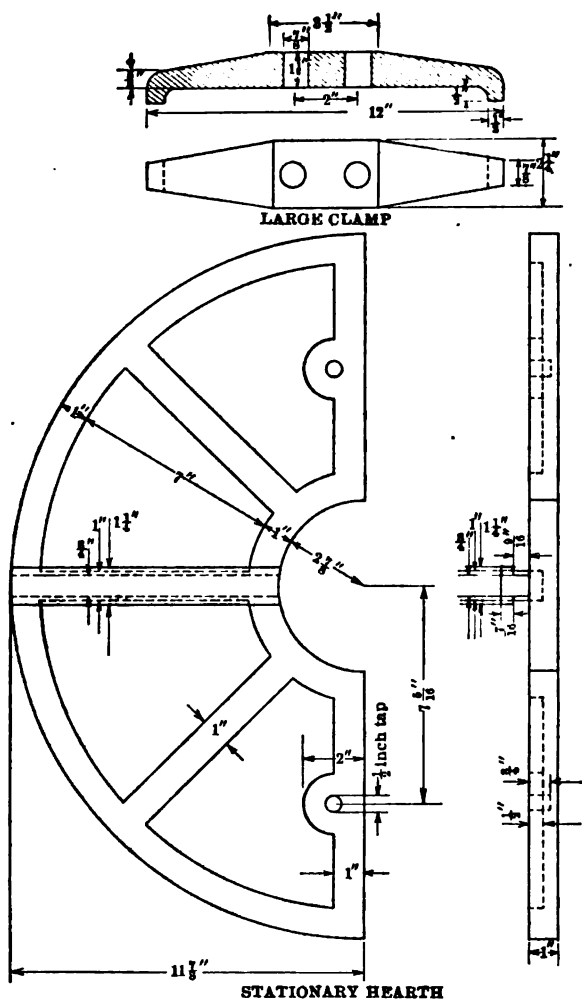


FIGURE 3.—Details of a stationary hearth in multiple-hearth furnace.

hearth to the outside of the top revolving hearth, falling onto the second hearth, a stationary hearth. It was then moved by rabbles attached to the top movable hearth to the center of the stationary hearth and fell onto the third hearth, a moving hearth. The ore was thus moved through the furnace in a manner similar to that in a regular large-size multiple-hearth roasting furnace.

dioxide. With these facts in mind, a study of the summary contained in Table 17 following, showing the reduction products, is interesting.

TABLE 17.—*Results of reduction tests with crucible involving eight cycles.^a*

Cycle.	Material reduced.	Per cent of total Ba present as—			
		BaS readily soluble in H ₂ O.	Insoluble in H ₂ O, soluble in acid.	Insoluble BaSO ₄ .	Reduced product No.
First.....	Pure BaSO ₄ by C.....	88.6	2.1	9.3	1
Second....	Precipitate formed by adding product 1 to SO ₂ solution.	91.3	3.9	4.8	2
Third.....	Precipitate formed by adding product 2 to SO ₂ solution.	90.3	5.4	4.3	3
Fourth....	Precipitate formed by adding product 3 to SO ₂ solution.	90.2	3.8	6.0	4
Fifth.....	Precipitate formed by adding product 4 to SO ₂ solution.	85.5	7.3	7.2	5
Sixth.....	Precipitate formed by adding product 5 to SO ₂ solution.	82.8	10.2	7.0	6
Seventh...	Precipitate formed by adding product 6 to SO ₂ solution.	77.9	15.0	8.1	7
Eighth....	Precipitate formed by adding product 7 to SO ₂ solution.	75.0	17.0	8.0	8

^a Tests by W. Freeman and V. P. Edwards.

CONVERSION OF OXIDE AND CARBONATE IN REDUCED PRODUCTS TO SULPHITE OR SULPHATE.

From the results of tests made in cycles it was evident that in order to conduct the process most successfully it would be necessary to do one of two things—either to effect the reduction in an indirectly fired furnace, such as a muffle or a retort, whereby there would be formed the minimum amount of the water-insoluble barium compounds, or, if the reduction was effected in a direct fired furnace, to introduce some method of converting these barium compounds back to the sulphite or sulphate for their subsequent reduction to sulphide.

Some experiments were made to determine the conditions required for the conversion of the oxide and the carbonate to the sulphide by a reaction with sulphur dioxide in solution.

A typical experiment is summarized here to show the rate at which this conversion takes place.

A reduced product from the fifth cycle, from which the readily soluble barium sulphide and oxide had been leached out, was used. The residue contained 16 per cent of the barium as sulphide, 55 per cent as oxide, and 13 per cent as carbonate and silica, 4 per cent as sulphite, and 12 per cent as sulphate.

This material, pulverized to 100 mesh, was shaken vigorously for 20 minutes with a solution containing 9 grams of sulphur dioxide per liter. After filtration there was still excess sulphur dioxide in solution. The residue contained a trace of barium sulphide, and 40 per cent of the barium was still present as oxide. Considerable barium (not determined) had gone into the mother-liquor solution.

Another sample was pulverized to 200 mesh and shaken for one hour with a solution containing 6 grams of sulphur dioxide per liter. The sulphur dioxide present was in excess of that required for reaction with the barium sulphide and oxide in the sample to form normal sulphite. After the hour's shaking about 26 per cent of the barium had gone into the solution, and the residual barium still contained considerable, about 30 per cent, barium oxide.

From this and other similar experiments it was determined that there was little chance for converting these compounds to the sulphite in the regular precipitation process, unless the solutions were kept strong in sulphur dioxide or the time allowed for agitation was much longer than could be considered practicable.

Thus it was decided that in order to keep these compounds in the cycles it would be necessary to leach out the water-soluble compounds occasionally and treat the residue with fairly strong sulphur dioxide solutions in a separate apparatus.

Tests were made in which the water-insoluble residues were treated in an agitator or pebble mill with sulphur dioxide solutions of different strengths for varying periods of time. Stated briefly, the tests showed that these insoluble barium compounds when pulverized finely can be converted into sulphite and sulphate by agitation with a sulphur dioxide solution of about 1 per cent strength for several hours. When the action was facilitated by further grinding and agitation, as in a pebble mill, the action was fairly rapid. In this treatment some of the barium oxide was dissolved as a bisulphite, so that to save the barium in practice this solution would have to be returned to the precipitation system.

Dilute sulphuric acid also converted these barium compounds to the sulphate.

GENERAL SUMMARY AND CONCLUSIONS.

The results of the investigations herein described warrant the statement that every step of the wet Thiogen process can be controlled and carried out successfully on a small laboratory scale, and the indications are that the same success can be expected in the large-scale operations of a commercial unit. There are no factors to prevent the successful conduct of the following operations: Absorption of the sulphur dioxide gas by the mother liquor solutions, precipitation of the sulphur compounds by the addition of barium sulphide to the solutions, settling of the precipitate, decantation of the solution, dewatering or filtering of the precipitate, and distillation of sulphur from the dried precipitate.

In the reduction operation also there are no factors that prohibit the conduct of the process. It is true that in all reduction methods some sulphur is lost, resulting in the formation of oxides or carbonates of barium that react slowly in the precipitation operation. However,

the proportion of these insoluble barium compounds can be reduced to a low figure by rapid reduction with carbon at a high temperature in a furnace externally heated. If reductions are effected in a direct-fired furnace, the proportion of these compounds is greater. However, as these insoluble compounds increase in the cycles, the soluble barium compounds can be leached out and the insoluble compounds allowed to react with a strong sulphur dioxide solution, resulting in their conversion to sulphite and sulphate, which can be reduced to the sulphide.

Regarding the commercial feasibility of the process, the following points must be considered:

In the absorption operation it is necessary that the percentage of sulphur in the gases should be as high as possible. It seems probable that a gas concentration of less than 6 per cent should not be considered in practice. Furthermore, a fairly constant concentration of sulphur dioxide in the gases is necessary if concentrated solutions and a high absorption of the sulphur dioxide are to be attained at the same time.

The relative solubility of sulphur dioxide in solution at different temperatures must be considered also. At 33°C. (91° F.), the volume of solution required to absorb a given weight of sulphur dioxide would be about twice that at 11° C. (52° F.). Unless special cooling devices were introduced, the temperature of the mother liquor would be nearly that of the surrounding atmosphere, certainly not lower. Thus the cost of absorption would depend to a certain extent on the average temperature of the neighboring air.

The gas should also be cleaned of dust and fume before it is sent to the absorption tower, otherwise the solution will become foul with the soluble compounds and the precipitates will become contaminated with flue dust and other impurities.

The operations of precipitation, settling, decantation, filtering, drying, and distillation called for in this process should not be any more difficult to conduct on a commercial scale than similar operations now carried on in standard commercial apparatus.

However, the efficiency of the settling, decantation, and filtering operations are largely dependent on the conditions under which the precipitation takes place. Thus, it is necessary that the sulphur dioxide content of the solutions coming from the absorption tower should be fairly constant, or the constant attention of an attendant will be necessary to see that the solutions are not too acid or alkaline after precipitation. This part of the process will probably require careful supervision.

In the commercial operation of the reduction process the tendency for the formation of insoluble barium products must be considered an important factor.

To reduce the amount of these insoluble compounds to a minimum the reduction must be effected rapidly, with carbon at a high temperature, and out of contact with the furnace gases. This would necessitate reduction in a muffle type furnace. At the high temperature required, such reduction would be more expensive to carry on than a reduction in a direct-fired furnace, as for example in a multiple-hearth furnace or in a cement kiln.

Even with this type of reduction, in a commercial application of the process, there would be a gradual accumulation of these "inactive" barium compounds, which would require special treatment with strong sulphur dioxide solutions or dilute sulphuric acid for conversion to compounds that could be reduced to the sulphide.

If the reduction is to be effected in a direct-fired furnace the necessity will be still greater for a supplementary operation to convert the barium oxide or carbonate to the sulphite or sulphate. However, by the introduction of this conversion operation the loss of barium as insoluble compounds through a series of cycles can be reduced to a low figure.

Another possible method of handling some of these water-insoluble but acid-soluble barium compounds would be to dissolve them in dilute acid after the water-soluble sulphide and oxide had been leached out, and thus prepare barium salts for the market.

Reducing the precipitate in a direct-fired furnace would involve in each cycle a loss of at least 5 per cent of the barium as dust, unless the dust was precipitated by the electrical precipitation or a similar process.

The results of the laboratory study of the wet Thiogen process demonstrate that the technical operation of the process can be carried out successfully for the recovery of sulphur from sulphur dioxide in waste smelter gases, and the indications are that at least in some localities the process can be applied on a commercial scale and sulphur recovered at a cost that will allow a profit.

ACKNOWLEDGMENTS.

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The writer also wishes to express his appreciation of the ready cooperation and valued counsel of Dr. L. H. Duschak, chemical engineer, Bureau of Mines.

APPARATUS REQUIRED AND MATERIAL HANDLED TO PRODUCE 10 TONS OF SULPHUR A DAY.

Based on the results of the critical study of the Thiogen process herein described, an outline has been prepared showing the apparatus required and the approximate quantities of material to be handled in the production of 10 tons of sulphur a day by the wet Thiogen process.

In this outline it is assumed that—

The furnace gases contain 7 per cent SO_2 .

The furnace gases are delivered at the absorption tower at 20°C . and free from dust and fumes.

The temperature of the mother liquor can be maintained at 20°C . or less.

An absorption of 80 per cent of the total sulphur dioxide in the gases is satisfactory.

Reduction is effected in a cement kiln.

Dust from the cement kiln is recovered by the electrical precipitation apparatus.

In the precipitation process no attempt is made to effect a reaction between the comparatively insoluble barium compounds and the sulphur dioxide by allowing excess sulphur dioxide to be present in solution after the readily soluble compounds have reacted. These insoluble compounds are to be converted to the sulphite or sulphate by a supplementary process.

DETAILS REGARDING PARTS OF APPARATUS.

Prescribed details regarding parts of the apparatus are given below in semitabular form.

ABSORPTION TOWER.

Total volume of gases to tower per minute at 20°C ., 2,650 cubic feet.

Total SO_2 in 2,650 cubic feet, 31 pounds per minute.

SO_2 absorbed in tower, 25 pounds per minute.

If the tower is constructed on the same lines as was the experimental tower, it would have the following dimensions: Total area of cross section, 70 square feet; area of space through the grill work, 25 square feet; total height, 50 feet.

The absorption height may be provided by single tower, or, as planned by the Thiogen Co., by three towers, with the circulation of part of the solution through the middle tower to build up the concentration. The length of time of contact of the gases and the solution would be about the same in either case.

The volume of mother liquor at 20° C. required to absorb this weight of sulphur dioxide would be about 38 cubic feet, or 277 gallons. The resulting solution would contain 25 pounds of SO_2 .

REGULATING TANK.

Mother liquor to flow by gravity from absorption tower to regulating tank.

Tank to be covered to prevent loss of SO_2 .

Dimensions: Diameter, 8 feet; height, 4 feet; capacity, 100 cubic feet, or 748 gallons.

PUMP.

Pump to force 277 gallons of solution per minute from regulating tank to agitator.

INCORPORATOR AND MIXER.

An automatic incorporator, as, for example, an adjustable worm-screw feed, would be required by which the proper proportion of the reduced material could be added to all, or a part, of the mother liquor as it is pumped from the regulating tank to the agitating tank. If the sulphur dioxide concentration of the solution can be maintained nearly constant, such an automatic incorporator would not require much close attention on the part of attendants. Centrifugal pumps are effective mixers.

Assuming that the average reduced product to be incorporated contains not less than 70 per cent BaS , 63 pounds of reduced material containing 44 pounds of BaS plus 19 pounds of other compounds not reacting with SO_2 are to be incorporated per minute.

AGITATING TANK.

The agitating tank, such as a Dorr agitator, should have a capacity of about 1,200 cubic feet (9,000 gallons), or the equivalent of about 30 minutes' supply of solution. Agitation should be vigorous and accomplished by mechanical means, as by paddles, rather than by compressed air. The overflow from the agitator should flow to settling tanks.

SETTLING TANKS.

Two tanks in series, each with a capacity of about 1,000 cubic feet (7,450 gallons), or a total settling capacity equivalent to 50 minutes' supply of solution.

Tanks should be equipped with slowly revolving rabbles, arms and rabbles to move precipitate to a center discharge at bottom.

A settled precipitate containing 50 per cent water may be obtained. In fact, the precipitate may be so thick as to require a special pump for moving it to a filter.

Total weight of precipitate to be settled per minute, 88 pounds, or 63.5 tons a day.

Mother liquor flowing to pump to be returned to absorption tower by pump; will contain some light, flaky sulphur, which will not clog in the pump nor settle in the tower.

FILTERS.

Filters must handle daily 126 tons of settled precipitated sludge containing not over 50 per cent water. With a vacuum of 18 inches a 1½-inch cake can be obtained in three minutes. Filtered cake should contain less than 25 per cent water. Discharge from filter should go into conveyor to be run onto drying furnace.

PUMPS.

Pumps to handle 250 to 270 gallons of mother liquor per minute from the overflow of settling and from the filter to the top of the absorption tower—approximately a 60-foot head.

DRYING FURNACE.

Material to be dried daily, 84.5 tons containing a maximum of 25 per cent water. Because of the necessity of drying at comparatively low temperatures, a steam drier

would probably be necessary. It is estimated that the gases from the reduction furnace, if burned under a boiler, should furnish sufficient heat energy to carry on the drying operation.

DISTILLING FURNACE AND CONDENSER.

Sixty-three tons of dried precipitate per day must be heated to 450°C . in a distilling furnace, and 10 tons of sulphur distilled. For this purpose a muffle-type furnace, equipped with rabbles or other stirring devices, may be used. A revolving cylindrical retort also might be employed. The circulation of a stream of sulphur dioxide through the distilling furnace into the condenser will facilitate the distillation of sulphur.

The daily fuel consumption, unless waste heat were available, would probably be 3 to 4 tons of coal, or 12 to 16 barrels of oil, depending on the construction of the furnace. However, the waste heat from the reduction furnace can also be utilized for the distillation and the actual fuel consumption would be much less. Coke or coal could not be mixed with the material to be heated in the distilling furnace, for there would result the formation of some carbon-sulphur compounds, or compounds of sulphur and hydrocarbons, which would reduce the purity of the sulphur condensed.

The residue consists of barium sulphate, sulphate, oxide, carbonate, and possibly some sulphide. It is pulverulent and is ready for mixing with the pulverized coal or coke.

PULVERIZER FOR COAL OR COKE AND MIXER.

Coke or coal, pulverized to at least 160 mesh, or lampblack carbon must be mixed intimately with the residue. Fifteen per cent fixed carbon should be provided. Mixing may be accomplished in a tube mill, as the material is taken from the distilling furnace and is then sent directly to the reduction furnace, thus saving heat.

Residue from distilling furnace, 53 tons. BaSO_3 and BaSO_4 present, 45 to 48 tons. Carbon to be mixed, 6 tons of coke, or 13 tons of coal.

REDUCTION FURNACE.

The daily charge to the reduction furnace would be 53 tons of barium compounds, containing 45 to 48 tons of BaSO_3 plus BaSO_4 , with 6 tons of carbon, as coke, or 13 tons as coal.

The maximum temperature desired in the furnace would be about 1050°C . In order that the reduction be accomplished most effectively, the material should be heated rapidly to 850°C . or higher and should be put through the furnace in a short time.

Rather than realize in the reducing furnace the maximum heat value of the fuel burned in the furnace it would probably be preferable and most economical to allow the furnace gases to leave the reducing furnace while still very hot, that is, about 500°C . This waste heat could then be utilized in the distilling furnace or in a boiler, and at the same time any combustible gases present in these gases could be burned to furnish steam for the drying operations.

A cement kiln about 70 feet long and $7\frac{1}{2}$ feet in diameter would probably handle the material in 24 hours. For this purpose the fuel consumption would be about 6 tons of bituminous coal with a heating value of about 12,000 British thermal units per pound, or 20 barrels of crude oil per day.

If reduction was effected in a muffle type furnace, the fuel consumption would undoubtedly be almost twice as great as in a direct-fired furnace. If estimates of the fuel consumption are based on the current practices of muffle-furnace reductions, a fuel consumption of one-third of a ton of coal per ton of material reduced would be very low. This would mean 15 to 16 tons of coal per day for carrying on the reduction in the muffle furnace.

ELECTRICAL PRECIPITATION APPARATUS.

Electrical precipitation apparatus, to handle approximately 10,000 cubic feet of gas per minute at 450° C., and to recover about 2 tons of dust per day.

PULVERIZER.

The reduced material, 35 to 40 tons, must be pulverized to at least 80 mesh before it can be used as a precipitant satisfactorily. The finer the material is ground, the more readily will the comparatively insoluble products react with sulphur dioxide in the solution.

LEACHING TANK.

It would be necessary to provide for leaching the water-soluble compounds from the reduced material whenever the proportion of the barium sulphide was reduced to 60 or 65 per cent of the total. If the reductions are effected in a cement kiln it would probably be necessary to leach 4 or 5 tons of material a day. The solution of barium sulphide would have to be returned to the agitation tank or to the precipitation tank.

AGITATING TANK.

Agitating tank for converting barium oxide to sulphite or sulphate, to handle 3 or 4 tons a day.

This tank would have to be covered to prevent the escape of sulphur dioxide gas. A ball or pebble mill would be effective for this operation.

CONSTRUCTION COST.

An estimate of the cost of construction of a 10-ton plant equipped as above outlined, with all the necessary accessories, based on the cost of material, etc., in 1913, indicates that the cost would be about \$40,000.

ESTIMATED COST OF PRODUCTION.

An estimate of the cost of production of sulphur from such an installation producing 10 tons a day has been made, based on costs of material and labor as follows:

Costs of material and labor used as basis for estimate of cost of production of sulphur from 10-ton plant.

Fuel:	
Coal, per ton.....	\$3.00
Oil, per barrel.....	.80
Reducing agent:	
Coal, per ton.....	3.00
Lampblack, carbon, or coke, per ton.....	6.00
Barium sulphate, per ton.....	7.50
Power, per horsepower hour.....	.007
Labor:	
Common, per day.....	2.50
Skilled, per day.....	3.50

The detailed estimate of the cost of production of sulphur follows:

Detailed estimates of cost of production of sulphur from 10-ton plant.

Power, including power for pumping solutions and precipitate sludge, for operating incorporator, agitator, filter, drying furnace, pulverizer, mixer, and reduction kiln, and for causing gases to circulate through absorption towers.....	\$6.00
Fuel and reducing carbon.....	60.00
Labor:	
1 skilled operator per shift at \$3.50.....	\$10.50
2 attendants per 8-hour shift at \$2.50.....	15.00
1 furnace man per shift at \$3.....	9.00
	34.50
Barium sulphate, unavoidable loss.....	5.00
Interest (6 per cent) and depreciation (including repairs), 16.5 per cent.....	24.50
Total.....	130.00
Cost per ton of sulphur.....	13.00

With a larger plant, some of these items, especially that of labor, would be somewhat decreased per ton of sulphur produced. With a 20-ton plant, the cost would be about \$12 per ton.

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FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

THE USE OF MUD-LADEN FLUID IN OIL AND GAS WELLS

BY

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AND

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THE USE OF MUD-LADEN FLUID IN OIL AND GAS WELLS.

INTRODUCTION.

The Bureau of Mines is investigating the technology of petroleum and its products, the investigation including well-drilling methods and the transportation, treatment, and utilization of petroleum and of natural gas, with especial reference to the prevention of waste and to increased efficiency in the development of oil and gas resources in land belonging to or controlled by the Government.

During the winter of 1912-13 the enormous waste of the gas encountered while drilling for oil in the Cushing field, Oklahoma, called forth protests from various sources, and appeal was made to the President of the United States and to the Secretary of the Interior in behalf of the landowner and of the public in general.

The facts were wanted regarding the amount of waste and the technical conditions that were responsible for practices causing it.

An investigation was ordered and conducted under the direct supervision of the late J. A. Holmes, the first Director of the Bureau of Mines. R. S. Blatchley was sent to investigate the amount of waste going on in the Mid-Continent fields.^a In the latter part of March, 1913, J. A. Pollard and A. G. Heggem were sent to Oklahoma to study well-drilling methods with reference to the waste of gas from wells on Indian lands and to suggest means of preventing the waste of natural gas on such lands.

Pollard and Heggem decided that the mud-laden fluid process, which had been used successfully for some years in Texas, Louisiana, and California was the method that would meet the situation most efficiently and economically. The method was accordingly demonstrated in Oklahoma.

In 1914 Congress provided for the employment of six oil and gas inspectors to supervise oil and gas mining operations on lands of the Five Civilized Tribes in Oklahoma and to conduct investigations with a view to the prevention of waste. Positions under this appropriation have been held by William F. McMurray, James O. Lewis,

^a Blatchley, R. S., Waste of oil and gas in the Mid-Continent fields: Tech. Paper 45, Bureau of Mines, 1913, 54 pp.

Harry D. Aggers, Ralph R. Weed, George W. McPherson, James C. Fowler, Louis W. Courtney, and Carl H. Beal, the technical work of these men being directed by W. A. Williams, chief of the petroleum division, Bureau of Mines. After careful and impartial investigation of conditions in Oklahoma, these men also decided that the mud-laden fluid process was the one best adapted to prevent further wastes.

In 1915 laws were passed by the State of Oklahoma that provide for the conservation of natural gas and oil, authority being given to the corporation commission to make rules and regulations and to employ agents to enforce them. After hearings and investigations, the corporation commission reached substantially the same conclusions as the Federal agents regarding the importance of the wastes and the best methods of conserving the gas.

Technical Papers 66^a and 68^b were published by the Bureau of Mines in December, 1913, and January, 1914, respectively. Technical Paper 66 deals with the mud-laden fluid process in general and No. 68 with its application in Oklahoma. The interest aroused by these publications has been so widespread that within two years the first editions have been practically exhausted. Rather than reprint these publications, it has been considered advisable to replace them with the present report, which includes the results of more recent investigations.

The methods of using the mud-laden fluid are an outgrowth of the rotary drilling system, which was developed in Texas and later employed in Louisiana and California. Some of the principles of the use of mud fluid had been observed and used to a limited extent for a long time in cable-tool drilling. So far as known it was first used successfully with a circulator and cable tools in the Coalinga field, California, in 1906, Harry D. Aggers, one of the contributors to this paper, working on the well. The part of the Bureau of Mines has been to call attention to its merits and to demonstrate its practical use in fields like those of the Mid-Continent.

ACKNOWLEDGMENT.

This report was written in collaboration with J. A. Pollard, A. G. Heggem, H. D. Aggers, W. A. Williams, L. W. Courtney, G. W. McPherson, C. H. Beal, and J. C. Fowler, whose connection with the investigation covered by this report has been mentioned above. Material from Technical Papers 66,^c 68,^c and 130^d of the Bureau of

^a Pollard, J. A., and Heggem, A. G., Mud-laden fluid applied to well drilling: Tech. Paper 66, Bureau of Mines, 1913, 21 pp.

^b Heggem, A. G., and Pollard, J. A., Drilling wells in Oklahoma by the mud-laden fluid method: Tech. Paper 68, Bureau of Mines, 1914, 27 pp.

^c See footnote references above.

^d McMurray, W. F., and Lewis, J. O., Underground wastes in oil and gas fields and methods of prevention: Tech. Paper 130, Bureau of Mines, 1916, 31 pp.

Mines has been used freely. In addition to the information gathered by the Bureau of Mines and the oil and gas inspectors in the Five Civilized Tribes, information of value has been acquired from the gas conservation officers for the State of Oklahoma and from many individuals, associations, and companies.

WASTES OF NATURAL GAS.

The Bureau of Mines since its inception in 1910 has directed much effort toward informing the public of the importance of the wastes of natural gas and toward finding preventive measures. In order to convey an idea of the necessity of more efficient methods of utilization, a few cases of the waste of gas are cited.

In the Findlay field, Ohio, the Karg well alone wasted not less than 1,500,000,000 cubic feet.^a

The Minshall gas well, drilled in the Macksburg field of Ohio, was allowed to blow into the air for a period of 10 years. It was put under partial control in 1885, when it had a measured volume of 4,500,000 cubic feet daily and a rock pressure of 425 pounds. It was not entirely closed in until two years later. The total volume wasted could not have been less than 15,000,000,000 cubic feet, with a fuel value equivalent to 750,000 tons of coal.^b

White^c speaks of waste in Kentucky and West Virginia as follows:

From one well in eastern Kentucky there poured a stream of gas for a period of 20 years without any attempt to shut it in or utilize it, the amount of which it has been figured was worth at current prices more than \$3,000,000.

In my own State of West Virginia only eight years ago not less than 500,000,000 cubic feet of this precious gas was daily escaping into the air from two counties alone, practically all of which was easily preventable by a moderate expenditure for additional casing.

The commission on the conservation of natural resources in Louisiana found that at one time 75,000,000 cubic feet of gas was wasting daily in the Shreveport field, one well alone wasting \$2,000 worth. Arnold and Clapp state that at certain times as much as 400,000,000 cubic feet has been wasted daily in the Caddo field, Louisiana, and that two wells with an estimated volume of 20,000,000 to 30,000,000 cubic feet daily burned for several years without restraint.^d

In drilling for oil in the Buena Vista Hills field, California, one gas well was opened that wasted at least 55,000,000 cubic feet per day for three months before it was finally got under control. At least 5,000,000,000 cubic feet of gas, valued at 25 cents per 1,000 cubic feet, or \$1,250,000, was lost from this one well alone.^e

^a Orton, E., and others, *Economic geology*: Ohio Geol. Survey, vol. 6, p. 133, 1888.

^b Statement of Orton Dun, Marietta, Ohio.

^c White, I. C., Address at conference on the conservation of natural resources, May 13, 1908.

^d Arnold, Ralph, and Clapp, F. G., *Wastes in the production and utilization of natural gas and means for their prevention*: Tech. Paper 38, Bureau of Mines, 1913, pp. 8-9.

^e Arnold, Ralph, and Clapp, F. G., *Op. cit.*, p. 10.

One well in the Petrolia field near Wichita Falls, Tex., flowed unrestricted for 11 months, during which time the loss is estimated to have been approximately 4,000,000,000 cubic feet.^a

Blatchley estimated that the waste of gas in the Mid-Continent fields to the end of 1912 would be 425,000,000,000 cubic feet as a conservative estimate.^b The heating value would be equivalent to approximately 21,250,000 tons of coal.

One well in the Osage Reservation, Okla., was allowed to blow for a year and a half, at the end of which time it was shut in by the use of mud-laden fluid and was found to gage 12,500,000 cubic feet daily from one sand alone. The amount of gas which escaped to the surface could not have been less than 6,000,000,000 cubic feet. The total waste of gas from this and other sands, both to the surface and underground, was far greater.

A shallow gas well, 520 feet deep, located in the Cushing field, Oklahoma, wasted 1,500,000,000 cubic feet of gas in 50 days. Practically every deep well in this field wasted many millions of cubic feet of gas while drilling. Wells wasting 25,000,000 cubic feet or more daily were numerous. At times the waste exceeded 500,000,000 cubic feet daily. In still weather the field was overhung with a haze of gas and oil vapors, and the amount of gas in the air was so great that no fires, open lights, or automobiles were allowed among the wells. Much damage was done and many lives were lost by the ignition of gas which had traveled many hundreds of feet from its source. Probably the waste in the Cushing field has never been exceeded in any other field. In May, 1915, an estimate was made by W. F. McMurray and H. D. Aggers, United States oil and gas inspectors, which showed that 207,000,000 cubic feet of gas was wasting into the air from 360 wells only. This was after the greatest waste had taken place, and included less than half of the wells. There is no way of determining the total waste, but for the three years ending in 1915, it was probably not less than 250,000,000,000 cubic feet; equivalent in fuel value to 12,500,000 tons of coal. For a period of many months, the rock pressure declined at the rate of one or two pounds a day. By far the largest part of this waste could have been prevented without difficulty by the use of mud-laden fluid, and it is evident that the saving of gas would have recompensed the operators many times over for the comparatively small additional expense.

These cases represent only a few of the more striking instances of waste in various fields of the United States. There are, also, thousands of instances individually of less importance, whose aggregate waste greatly exceeds that from the smaller number of con-

^a Blatchley, R. S., Waste of oil and gas in the Mid-Continent fields: Tech. Paper 45, Bureau of Mines, 1913, p. 43.

^b Blatchley, R. S., Op. cit., p. 47.

spicuously large wells. Frequently the amount of gas utilized has been but a small percentage of the content of the field, and it is believed to be no exaggeration to say that not half of the natural gas so far developed has been utilized.

WAYS IN WHICH NATURAL GAS IS WASTED.

Ways in which natural gas is wasted from wells for which mud-laden fluid may be employed as a preventive measure, are: (1) During the drilling of wells; (2) from behind casings; (3) from uncontrolled or wild wells; (4) from wells on fire; (5) through defective casing, packers, or casing seats; (6) by unsystematic ways of casing wells in a field; (7) by not separating gas sands from producing oil sands; (8) by migration from gas sands into other strata; (9) by the drowning out of gas by water from other strata; (10) from improperly plugged wells.

The larger part of the waste that has taken place in the past could have been prevented with the knowledge and equipment then available. However, some of the wastes could have been controlled only by the knowledge that has been gained in recent years. At present, cases where waste can not be prevented by reasonable foresight and expense are exceptional.

The greatest wastes of natural gas have taken place in fields which were being developed for oil. In most oil fields, much gas is found in sands above the oil or in the same sand as the oil. If a well is being drilled by one of the usual methods, the gas becomes a hindrance to drilling, and the driller regards it as a nuisance. Or the gas may be found in a field where it has little or no immediate commercial value, and hence is allowed to escape into the air without restraint. It has been a frequent practice to allow the gas to escape freely while the well is being drilled through the gas-bearing sands in search of oil (Pl. I, *A*), or the gas has been allowed to blow into the air for long periods, for the purpose of reducing the volume and pressure so that drilling may be conducted with less effort in later wells. Such great wastes are believed to be altogether unnecessary, for the preventive methods that are described in this paper have proved to be entirely practicable.

In a field where much gas overlies the oil, the usual dry-hole methods of drilling can not be employed without danger and waste. In the Mid-Continent fields alone during recent years there have been a large number of deaths and serious accidents from burning gas (see Pl. I, *B*) while wells were being drilled. These accidents and the great hazard from fire risk, to say nothing of the great waste of gas, could not have happened if the mud-fluid method were properly used.

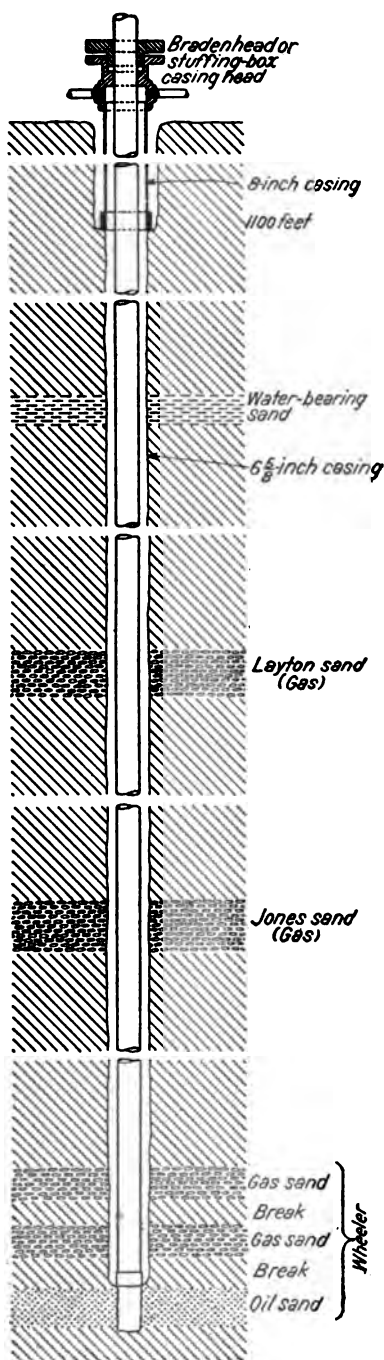


FIGURE 1.—Generalized section of well, showing open passage about casing.

To place a casing properly, the drill hole must be large enough to allow the couplings to slip freely down the hole. There is, therefore, a space of an inch or more between the casing and the walls of the hole (fig. 1). This makes a free path around the casing, which allows oil, gas, or water to pass freely from one stratum to another or to the surface. The water may drown out the oil or gas, the gas may escape into porous strata, thus reducing the volume and pressure until it is of no commercial value, or fresh water may be spoiled by oil or salt water. By the use of mud fluid this intercommunication behind the casing is prevented (fig. 2).

IMPORTANCE OF UNDERGROUND WASTE.

The importance of underground waste has not been given as much attention as the facts warrant.^a To prevent such waste it is necessary to confine closely oil and gas to the strata in which they are found, otherwise they will dissipate through the formations and may not be recovered (fig. 3). One of the most common arguments against shutting in gas has been that once a deposit has been opened up, the gas can not be held underground, but in some way escapes, and therefore any expense incurred in shutting it in is not warranted. If the gas is closely confined to its original stratum it can be held for future markets. It is believed that the use of the mud-laden fluid process will conserve the gas at expenses which the future returns will more than justify.

^a McMurray, W. F., and Lewis, J. O., *Underground wastes in oil and gas fields and methods of prevention*: Tech. Paper 130, Bureau of Mines, 1916, 31 pp.



B. A BURNING GAS WELL.



A. GAS WASTING FROM A WELL BEING DRILLED.

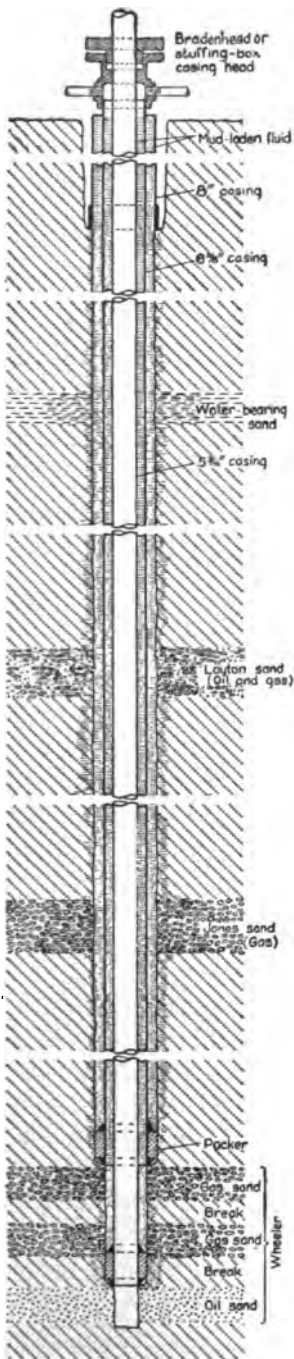


FIGURE 2.—Generalized section of "combination" well finished with mud-laden fluid.

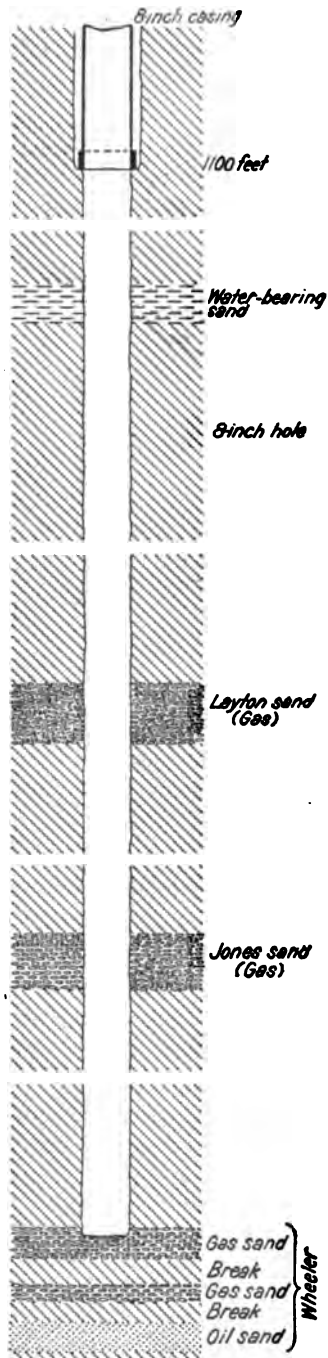


FIGURE 3.—Generalized section of uncased hole, permitting communication between formations.

To show that underground waste is a fact which has been established for many years, the following is quoted from the first report of the supervisor of natural gas in Indiana, issued in 1891:^a

I would also recommend the passage of a law regulating the packing of wells. A large number of wells have what is known as wall packers. By this is meant that the packer and tubing is anchored to the wall of the well in the shales, in many instances as much as four and five hundred feet from the bottom of the well. Incalculable injury is, in nearly every case, done by this method. The gas, after its release from the Trenton rock, permeates every portion of porous rock that forms the wall of the well below the point at which the well is packed. It spreads itself laterally through every crevice and opening that is found, and thus an enormous waste is maintained.

In many parts of the field we find gas escaping through the earth and bubbling up through the water in the streams and in water wells. In some instances the escape into water wells has been so strong that the wells have been closed in and the gas piped into the houses adjoining and found sufficient for domestic consumption.

All this may be attributed to the insufficient packing of the wells in the vicinity. The great upheaval of earth and rocks, and the explosion which followed in the vicinity of Waldron, Shelby County, which occurred August 11, 1890, was due to the gas escaping through the shales and below the packers from the wells at Waldron and St. Paul. The gas escaping laterally through the shales collected in subterranean reservoirs until the pressure became so great as to cause the upheaval.

The waste from this cause has resulted in vast injury to the field in different localities by reducing the pressure, and the consequent introduction of salt water.

WAYS AND ADVANTAGES OF USING MUD-LADEN FLUID.

DESCRIPTION OF MUD-LADEN FLUID.

In this bulletin the term "mud-laden fluid" is applied to a mixture of water with any clayey material which will remain suspended in water for a considerable time and is free from sand, lime cuttings, or similar materials.

Some oil workers have thought "mud-laden fluid" implies the use of any of the drillings from the well, but this is not the case, for if the coarse materials in the drillings, such as sand, limestone fragments, etc., are left in the fluid they will settle and are likely to pack around the tools or the casing and cause serious troubles, specific instances of which are described in Bureau of Mines Technical Paper 68.^b The fine, sticky clays that are in many localities termed "gumbo" are well suited for the purpose, but clay or shale from other formations may be used, provided it is separated from the sand and other coarse materials and only the part that will remain sus-

^a Gorby, S. S., and others, Indiana, Geology and natural history: 17th Ann. Rept. State Geologist, 1891, p. 338.

^b Heggem, A. G., and Pollard, J. A., Drilling wells in Oklahoma by the mud-laden fluid method: Tech. Paper 68, Bureau of Mines, 1914, 27 pp.

pended in water is used. It is possible to use drillings from almost any formation containing clay or shale after proper treatment by settling.

CONSISTENCY OF FLUID TO BE USED.

The consistency of the fluid should be varied according to the conditions for which it is to be employed. Most frequently mixtures with a specific gravity of 1.05 to 1.15 are used in drilling—that is, 5 to 15 per cent heavier than water. When the fluid is not used to drill in, thicker fluid is often employed, which has the advantages of greater weight and of clogging up the pores more readily. Experience soon enables the operator to judge the consistency of fluid required for practical uses.

The operator who is unfamiliar with the use of mud-laden fluid is likely to use it too thin. This has been the cause of much trouble in Oklahoma. Such fluid acts like clear water. It will not clog the pores of the sand readily and hence will be forced into them for considerable distances, and in some instances near-by wells have been affected. It is also likely to cause caving and is injurious to the sand, or it may not have sufficient weight to overcome high-pressure gas. The only limit to the thickness of the fluid which it is possible to employ is whether or not it can be handled by the pumps, but it must be a fluid and not a pasty clay.

An idea of the consistency ordinarily required can be obtained by comparing the action of a stream of clear water with that of a stream of sand pumpings or muddy water running in a ditch. The sand pumpings contain clayey material which is deposited on the walls and especially the bottom of the ditch, where it forms an ever-thickening protective coating, whereas clear water cuts away the sides and bottom of the ditch and may cause it to cave. Between clear water and water containing more mud than it can hold in suspension, it is possible to find a mixture of clay and water that will deposit particles of clay as a fine protective coating, while the rest of the clay remains in suspension and passes through the ditch.

VARIATION IN FLUID PREPARED FROM DIFFERENT CLAYS OR SHALES.

There is considerable variation in the nature of fluids derived from different clays or shales. When the same volumes of two different materials are mixed with the same volumes of water the resulting fluids may differ considerably in specific gravity, or of two fluids of the same specific gravity one may be more viscous or thicker than the other.

SETTLING OF MUD FLUID.

An important consideration and one that has raised numerous inquiries is the amount of settling which a mud fluid will undergo. It is a well-known property of clays and similar colloidal materials that they will remain in suspension indefinitely. One sample of mud fluid has been standing for three years without appreciable settling and has not solidified. Numerous experiments conducted on short columns of fluid have shown that it will settle rapidly for a few hours, after which the rate of settling is very slow, and after a few days is practically nil. There is a surprising difference in what the maximum density may be. Of six samples collected from oil wells in southern California and allowed to settle two months, the variation was 10 to 30 per cent excess in weight over that of clear water. In each case the settled part was still fluid and not very viscous. It was also found that the final density in a short column of fluid is governed largely by the original density. For instance, two fluids were prepared by mixing the same kind of material in different proportions with water, one having a density 5 per cent greater than water and the other a density 15 per cent greater than water. The first settled to a much lighter consistency than the second.

When the fluid settles in a short tube it separates into clear or turbid water at the top and mud fluid at the bottom. The specific gravity of the fluid at the bottom varies in the manner outlined above. The proportion of water to that of the fluid which settled out depends principally on the specific gravity of the original mud fluid, and the lighter the original fluid the greater is the proportion of water to the settled fluid in the bottom of the tube. Although the settling takes place quickly in a short tube, the same rate of settling applied to a long column of fluid in a well means that it takes a very long time for it to settle, and in fact there is reason to suspect that behind the casing complete settling does not take place even after long periods of time.

When the fluid has settled for some time in a well there may be water near the top. As the weight of material in the well remains the same, the pressure at the bottom of the column of fluid is the same as before the settling occurred, but near the top the pressure will be decreased. However, the mud has sealed the formations, and no oil or gas can pass through unless the pressure within the strata is sufficient to force the mud out of the pores and to overcome the water pressure in the hole. When the fluid is to be left in the well it is advisable to use a heavy fluid that will settle very little. If there should be formations near the top of the hole which it is important to protect, fresh mud fluid can be put in after the original fluid has had a chance to settle.

All the evidence obtainable from practical experience points to the fact that mud fluid, properly mixed and employed, does not solidify behind the casing, even in deep wells and after long periods. This has been the experience in Texas and Louisiana with rotary wells, and in Ohio with wells drilled by cable tools.

DETRIMENTAL EFFECT OF COARSE MATERIALS IN THE FLUID.

Sand or other coarse materials in the fluid will ordinarily settle out, but if the fluid is too thick they may be held in suspension, and in such cases it has been found by experiment that the sand suspended in the fluid offers much resistance to anything being forced through. The action on a string of casing of sand suspended in the fluid is undoubtedly to increase the friction on the casing, and may be instrumental in making it so "loggy" that it can not be moved.

ACTION OF MUD FLUID ON POROUS FORMATIONS.

The action of mud-laden fluid in a sand or other porous formation can be likened to the action of muddy water going through a filter. In any filter that has been used for some time it will be found that most of the sediment from the water has been deposited on the surface of the filter, but some of it has entered the filter, the proportion diminishing with the distance penetrated. The distance to which mud from the fluid in the well will penetrate a porous formation depends partly on the combined pressure produced by the column of fluid and the pump, and partly on the consistency of the fluid and the porosity of the formation. At first the fluid will enter the formation, but finally the mud will clog the pores and no more water will go through. Ordinarily, if a thick fluid is used on the sands encountered in the well, it will not penetrate to any great distance even under high pressure, but if the fluid is too thin it may not clog the pores readily and will act more like clear water, which may enter a sand indefinitely. Occasionally a very coarse sand, a fissured formation, or a porous limestone is found into which even thick fluid may penetrate for some distance.

When no more fluid will enter the sand or porous formation a barrier or plug that is impervious to oil, gas, or water has been formed within the sand surrounding the hole. This plug is held in place partly by the resistance to movement of the mud deposited in the pores of the formation, but principally by the excess of pressure produced by the column of fluid in the hole. If the column of fluid is removed the pressure within the sand will usually force out the mud, and the oil, gas, or water will enter the hole again; but as long as a sufficient column of fluid remains in the hole the contents of the sealed formation can not enter the hole. It is believed that the

efficiency in sealing off the porous formations in a well depends more upon the mud forced into the pores of the formation and retained by the weight of the column of fluid than upon the mud plastered on the walls of the hole, although the mud coating probably aids in protecting the walls from caving.

When a well has been treated with mud fluid the contents of each formation is confined to its original stratum, so that there can be no movement of oil, water, or gas either from the sands into the well, from the well into the sands, or from one sand into another. Thus waste and intermingling are prevented, corrosive waters can not reach and attack the casing, and the strata are entirely sealed off from each other as they were before the well was drilled.

Mud fluid, besides preventing caving, as stated above, is also an aid in keeping loose sands from entering the hole. The fluid clogs up all pores or crevices, and makes a solid wall which the weight of the fluid in the hole will hold up. Furthermore, the mud which has entered the formations, or is plastered on the walls, protects them from contact with air and water, which would cause slaking and caving. The fluid is especially helpful in drilling through a loose sand that otherwise would run into the hole and make drilling difficult.

DISADVANTAGES OF USING CLEAR WATER.

Some drillers contend that clear water should have the same effect as the mud-laden fluid, but experience has shown that it does not. Many wells can not be filled with clear water, because the water continues to flow into the sand indefinitely with no clogging effect and consequently does not rise high enough in the well to give sufficient pressure to overcome that of the gas. Moreover, the action of the water on the walls of the hole causes caving, and an attempt to use clear water in a well invites trouble and may injure the producing sands.

The action of the mud fluid is entirely dissimilar to that of clear water. The fluid enters the porous stratum for a short distance and deposits mud, which clogs the openings and finally prevents further inflow of fluid. Clear water thus lacks some of the essential properties of mud fluid, such as the clogging effect and the greater weight.

PREPARING MUD-LADEN FLUID.

The mud-laden fluid may be prepared from clay obtained from surface deposits or from material derived from drillings. Ordinarily there will be enough clayey or shaley material in the formations encountered in the well to provide all the fluid necessary. This has been found true both in drilling with rotary tools and with cable

tools. In order to save time in preparing the fluid for use in the well, it is recommended that a slush pit, having a capacity of about 300 barrels, be constructed close to the drill rig, into which materials suitable for making mud-laden fluid may be run whenever encountered in drilling. Drillings from sandstones and limestones should not be allowed to enter the pit. The slush pit can be provided with

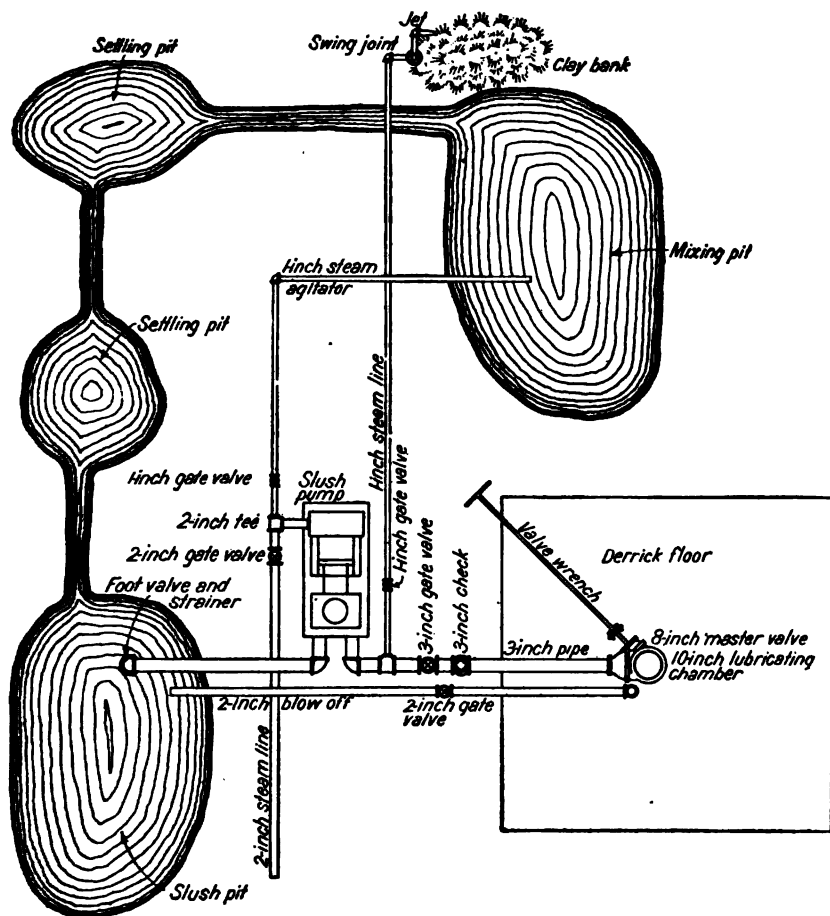


FIGURE 4.—Ground plan of equipment used in handling mud-laden fluid.

slight expense, and the mud fluid can be mixed and prepared in a few hours by ordinary unskilled labor whenever it is desired. Figure 4 shows one arrangement for a slush pit and settling pits. An arrangement for use with the circulator, a device for circulating mud fluid in a well, is shown in figure 5.

Settling out sand, limestone cuttings, etc., in order to avoid freezing of casing and of tools, is important.

METHODS OF INTRODUCING MUD-LADEN FLUID INTO THE WELL.

The method of introducing mud fluid into a well is varied according to the circumstances under which it is to be employed.

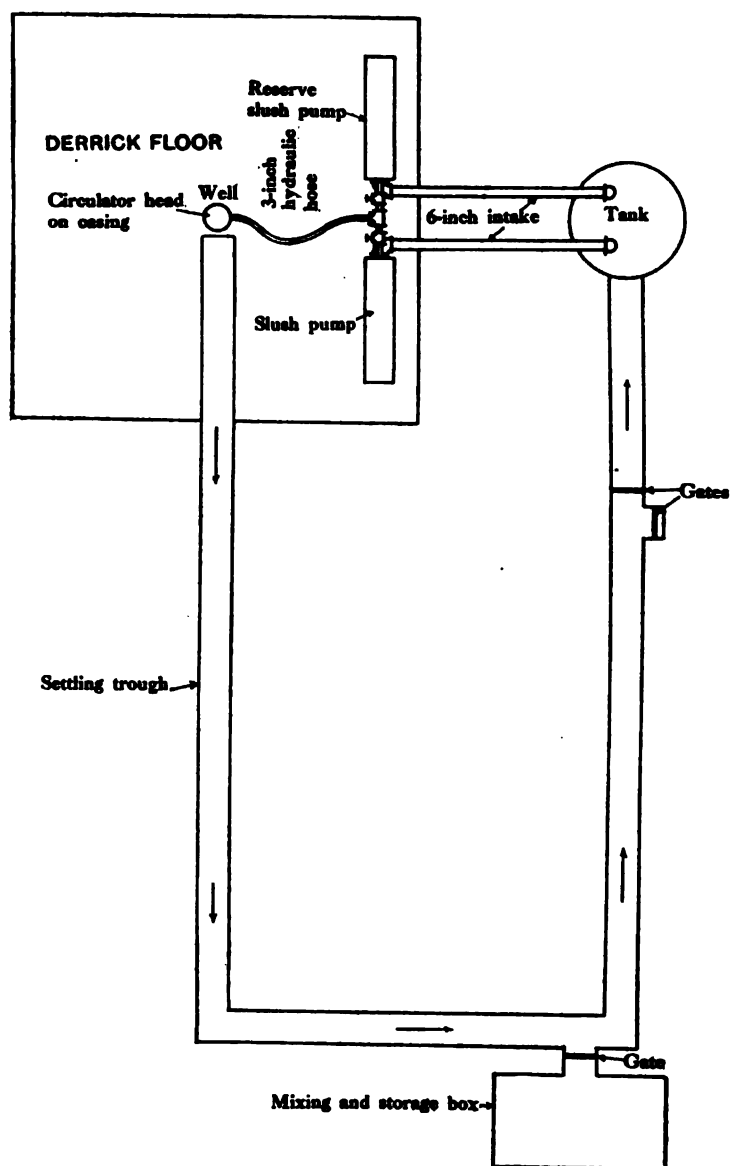


FIGURE 5.—Diagram of equipment for circulating mud-laden fluid in a well.

It is not advisable to let the fluid run down the walls of a dry hole because to do so may cause them to cave. If there is a considerable amount of open hole in a well, it should be filled by dumping the

fluid into the bottom of the hole with a bailer, or introducing it through a string of tubing or casing which reaches to the bottom. When high-pressure gas has been struck special methods must be employed.

THE LUBRICATOR METHOD.

If gas has been struck and the well is blowing gas, and the conditions are such that the gas can be shut in, recourse may be had to a method which has been named the lubricator system, one arrangement of which is shown in figure 6. The general features of this particular arrangement were devised by A. G. Heggem, who employed his control casing head instead of a master valve and tee. The apparatus consists of two joints of casing, preferably 10-inch casing, with a tee at the bottom, which in turn is connected by a nipple to a master valve placed on the head of the well. A 2-inch or 3-inch pipe leads from the slush pump into the tee at the bottom of the lubricator, with a check valve near the tee and an emergency gate valve near the pump. A 2-inch pipe leads from the top of the lubricator down the side, to which it is lashed, and thence to the slush pit, a gate valve being placed outside the derrick at some convenient point. The apparatus can be assembled and then placed on the well as a unit. In this manner the danger of handling single fit-

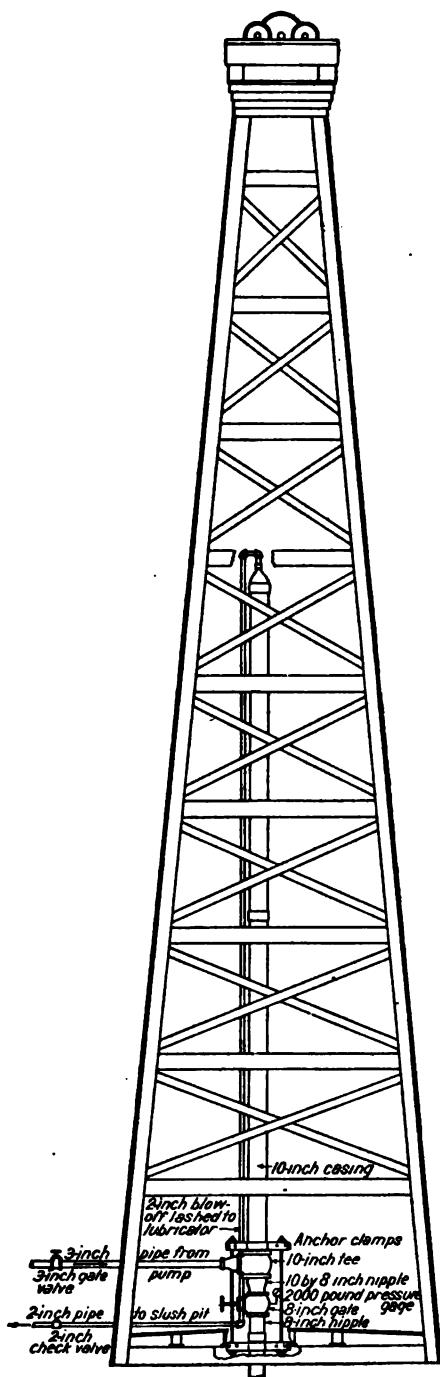


FIGURE 6.—Device for introducing mud-laden fluid into a well under gas pressure.

tings or putting a valve on a gas well is avoided. If a long crow-foot wrench is attached to the master valve so that it can be operated from without the derrick, there is ordinarily no necessity for any worker to be in the derrick while the apparatus is being used, thus reducing to a minimum the danger of shutting in a high-pressure gas well. After the lubricator has been set up the ordinary crew can operate it.

When the lubricator has been set up the master valve is closed and the valves on the pipe from the pump to the lubricator and on the blow-off pipe are opened. The mud-laden fluid is pumped into the lubricator until it is filled and fluid shows in the blow-off; then the blow-off valve is closed and the master valve is opened. The fluid then drops to the bottom of the well, being replaced in the lubricator by an equal volume of gas. After waiting two or three minutes for the lubricator to empty, the master valve is then closed and the blow-off valve is opened. The gas in the lubricator escapes through the blow-off valve, carrying into the slush pit the small amount of mud fluid that may have remained in the apparatus. The lubricator is then filled again, the blow-off valve is closed, and the master valve opened. These operations are repeated until the column of fluid in the well overcomes the gas pressure to such an extent that the pump can force the mud fluid in against the gas pressure and fill the well.

It has been found desirable in some cases to move the casing while lubrication is going on, in order to keep the pipe free. This can be done by handling the whole apparatus in slings, reaching from the collar below the master gate to the casing block above the lubricator. If the gas pressure in the well has not already been overcome by that of the fluid in the hole, the casing being filled must be worked up and down through a bradenhead or casing head of the stuffing-box type attached to the next larger string of casing; otherwise the fluid would be blown out between the casings when the inner casing was lifted off the seat.

PUTTING THE FLUID IN A WELL THAT CAN NOT BE SHUT IN.

When gas is blowing from a well and can not be shut in, perhaps because of the small amount of surface casing in the well, or possibly because the casing was not properly seated so that the gas is forced up outside of the casing when the valve is closed, the method shown in figure 7 is used. At such a well it becomes necessary to insert a string of tubing with a back-pressure valve placed on the bottom, or preferably a wooden plug, which is advisable because a back-pressure valve may get out of order at some critical moment. The wooden plug is driven lightly into the bottom of the tubing, so that it

can be forced out when desired, but is tight enough so that no gas can enter the tubing. After the tubing has been lowered to the proper depth it is packed or sealed with a casing tee previously placed on the well. To control the flow of gas, a gate valve should be placed on a lateral discharge of this tee. As soon as the mud fluid has started down the well through the tubing the gate valve on the tee can be partly closed in such a manner as to throttle the outlet and to prevent the mud-laden fluid from being forced out of the well by the gas pressure. The weight of the fluid when it reaches the bottom of the tubing will force out the wooden plug.

The amount of throttling necessary can be determined only by the man in charge, as similar conditions will not prevail at any two wells. However, it is not difficult to ascertain how much the gas should be throttled. If the well is emitting water with the gas, the fluid can be put in just as readily by this method, and a well with a capacity of as much as 40,000,000 cubic feet of gas and several thousand barrels of water daily has been quickly controlled. As is evident, the full rock pressure of a well is not maintained in the casing, and consequently no blow-out will follow, as would happen if the gas were forced to the surface on the outside of the casing.

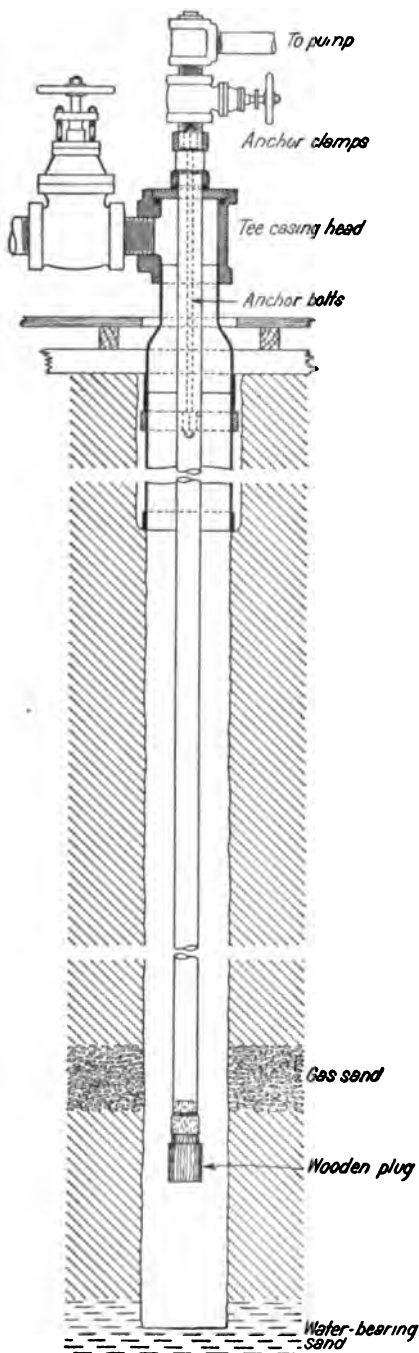


FIGURE 7.—Equipment for introducing mud-laden fluid into a well that can not be shut in. Well section is generalized.

OTHER METHODS OF INTRODUCING THE FLUID.

The throttling method can also be used to advantage instead of the lubricator, in wells that can be shut in, if the volume and pressure of the gas is not too great. The fluid is pumped in directly without use of the equalizing chamber. As care must be taken to prevent the internal pressure from splitting the pipe through which the fluid is to be introduced, it is not advisable to use light casing when it is to be subjected to very high pressure.

It sometimes becomes necessary to introduce the mud fluid between casings from the top of the well. This should be avoided if possible, for the reason that the fluid will carry down cavings and other material which may lodge against the couplings at some point in the hole and form a bridge which will keep the fluid from reaching the bottom, or will pack around the casing in such a way that it may not be possible to pull the casing should it ever be desired to do so. Experience shows that it is more difficult to pull casing that has been "mudded" in this way.

If the gas pressure in the well is not too high, a string of casing can be "mudded" by the following method, which, however, is not recommended, for the reasons given below. The casing is set below the gas sand and is then filled to the top with mud fluid. When the casing is raised, the fluid will rise outside the casing until a level is reached with that inside of the casing. The well is usually filled to a point between two-thirds and three-fourths of its depth, and the height of the fluid ordinarily is sufficient to overcome the gas pressure. The hole is then filled to the top, the casing is seated, and the fluid bailed out from the inside.

Although this is a very simple and quick method, it has serious disadvantages. If the well is drilled through gas, the gas is wasted while the sand is being drilled through and until the fluid is placed behind the casing. There is also great danger that the casing may freeze or collapse and can not be resealed so that it will hold in the fluid. In a well 1,000 feet deep, the pressure of the fluid at the bottom will be 500 to 600 pounds per square inch, and will increase in deeper wells proportionately. When the pipe is raised, the shoulder on which the casing rests and the walls of the hole have to bear the sudden rush of this large weight of fluid, and unless they are of very firm rock, the shoulder will be destroyed and the walls of the hole gouged out, which will make it very difficult to reseat the casing securely. Furthermore, the material loosened from the walls of the hole will be carried up between the casing and the walls of the hole for some distance, and then will settle back against the collars, and is likely to freeze the pipe, or it will be jammed suddenly under the collars one or two joints from the bottom with such great pressure as to

crush the casing. In two instances this method resulted in crushing the casing at the second joint from the bottom. In many others it was followed by trouble in seating or moving the casing.

PROCEDURE WHEN THE PRESSURE OF THE COLUMN OF FLUID IS INSUFFICIENT.

It occasionally happens that the pressure in a gas, oil, or water-bearing sand may exceed the pressure exerted by a column of fluid extending to the top of the well. When this happens, the fluid is blown out and the well goes "wild." It then becomes necessary to use a pump to overcome the gas pressure and to force the mud-laden fluid into the sand. Sufficient pressure should be allowed to remain on the well for at least two or three hours and then the pressure should be relieved slowly and carefully. If the pressure is suddenly reduced, the column of fluid may be violently ejected and the casing, fittings, and derrick wrecked. In some instances it has been found necessary to maintain the pressure for six hours or more.

SEALING OFF VERY POROUS OR COARSE-GRAINED FORMATIONS.

Generally when a sand is to be "mudded off," a comparatively thin mixture is first used, and this is followed up by a thick fluid. If the fluid is to be left in the well, it should be thickened so that there will be but little settling.

Occasionally a formation may be struck which is so porous and so coarse grained that difficulty will be experienced in clogging it with the fluid. If it can not be sealed off with the thickest fluid possible to use, it may be necessary to employ substances like sawdust, chopped straw or chopped rope, or other materials which will pack into the pores of the sand so that it will hold the mud. Trouble may arise from such material clogging the pump, but if necessary the lubricator can be employed in such a way that the material can be introduced without passing it through the pump.

TYPE OF PUMP TO BE USED WITH MUD FLUID.

A pump specially designed for handling thick mud fluid should be used. The type is generally known in the oil fields as a "slush pump" and is fitted with removable liners and insert valves. An 8-inch by 5-inch by 10-inch pump, weighing less than 2,000 pounds, has been successfully used on small wells in Oklahoma. Such a pump has the advantage of being inexpensive and easily moved. Where the gas pressure is high, a larger pump capable of greater pressure and of handling a larger volume of fluid is necessary. A 10-inch by 6-inch by 12-inch pump, weighing about 4,700 pounds, will meet anything but extraordinary conditions.

DRILLING WITH FLUID IN THE HOLE.

The presence of the mud-laden fluid in the well does not prevent drilling with cable tools, and the bailer can be used in the usual way for removing the drillings from the bottom of the hole. Drilling with cable tools in mud fluid has been a matter of ordinary practice in the California fields for many years both with and without the use of the circulator. Although it is not as fast as drilling in a dry hole, where conditions are favorable, there are places where the former method will have many advantages over the dry-hole method and sometimes may be faster and cheaper.

Drilling a well through a gas sand by the dry-hole method causes great waste. The free escape of gas has resulted in many disastrous fires and other damage to life and property. While the gas is escaping, no fires can be lighted near the well, so it is often necessary to stop work at night or in cold weather. Even work on wells near by may be stopped. Sometimes the gas pressure interferes with the tools dropping in the hole and makes drilling dangerous, slow, and occasionally impossible. Under such circumstances it may be necessary to discontinue drilling until the volume and pressure of the gas has been reduced. Either the well is allowed to blow without restraint or it is shut in and called a gas well, although there may be no market for the gas at the time. If, however, drilling is conducted with the hole full of mud fluid, these evils will not occur. The gas will be saved, danger prevented, drilling can be done at night and in cold weather without discomfort, and high-pressure gas will not interfere with nor prevent the working of the tools.

Occasionally strong flows of water interfere with drilling and may prove difficult to shut off. Mud fluid can be used to control the water, and in one instance was successful after several months' efforts to shut off the water with casing and packers.

In fields where wells have to be cased as fast as the hole is made, owing to caving, and where much underreaming is necessary, mud fluid can be used to much advantage. The fluid plasters up the walls of the hole, and the pressure exerted by the column of fluid usually will prevent caving or sand from running into the hole, which would interfere with drilling and underreaming or would "freeze" the pipe. It is possible with this method to carry the casing to much greater depths through a caving or sandy formation. In some Texas and California fields drilling without mud fluid, either with rotary or standard tools, would be practically impossible.

When drilling in mud-laden fluid through a rock, such as limestone, the cuttings from which are hard and coarse, great care should be taken not to drill too much hole at one time. Experience has proved that when too much hole has been made such drillings

will settle back around the tools and "stick" them in the well when the bull ropes are thrown on and the temper-screw clamps removed preparatory to drawing out the tools. This sticking of the tools is shown in figure 8. Trouble from this source was experienced in the demonstrations at the Greis-well and at the Sinclair, White & Ufer well in the Cushing field.*

From observations of engineers of the bureau, it is recommended that not more than 3 feet of hole should be drilled at any one time in such formation, when fluid is in the hole, unless experience has shown that this figure can be exceeded with safety.

The cuttings should be bailed from the bottom of the hole, and after they have been removed by running the bailer once or twice, it is good practice to bail the fluid from near the bottom of the hole and dump it into the top in order to keep the fluid of the same consistency from top to bottom. The fluid at the bottom of the hole is constantly being thickened from the drillings and by the settling of coarser materials which may not have been removed from the fluid. The fluid is thickened by drilling through clays or shaley formations but is apt to be thinned by drilling through sands.

The mud fluid used in drilling is generally 5 to 15 per cent heavier than clear water. In such a mixture of clay and water, the driller can scarcely distinguish the difference between the action of his tools from that in a hole full of clear water. It must be noted, however, that it can not be expected to drill in the same manner as in a dry hole, and much delay has been caused by failure to recognize this fact. As in a wet hole small tools should be used in order to provide greater clearance, that they may drop more rapidly through the fluid.

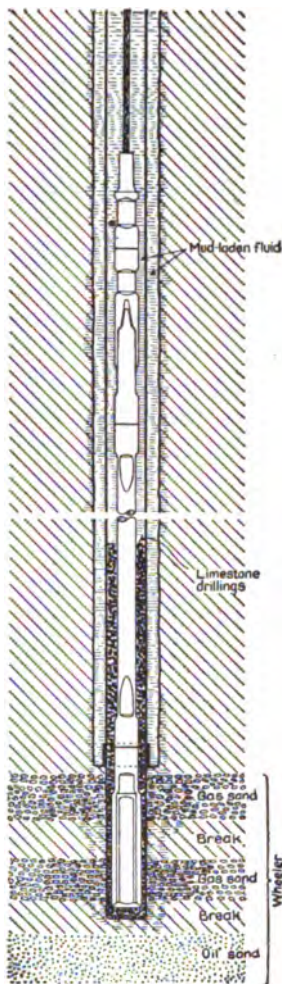


FIGURE 8.—A bit stuck by settling of drillings. Well section is generalized.

* Heggen, A. G., and Pollard, J. A., Drilling wells in Oklahoma by the mud-laden fluid method: Tech. Paper 68, Bureau of Mines, 1914, pp. 15-21.

It frequently happens when drilling through a gas sand, either with rotary tools or with cable tools and a circulator, that the mud fluid is mixed with gas and becomes so light that it will not hold back the gas, and may be blown out of the hole. When gas shows in the fluid in the well, it should be replaced with fresh fluid.

At times gas of such high pressure is struck that the fluid can not be maintained in the hole by its own weight, and it is necessary to stop drilling at intervals and shut in the well under pump pressure until the gas is driven back and the formation sealed off, then drilling may be resumed. Care should be taken, however, not to drill too deep at any one time without applying the necessary pump pressure, and precautions should be taken to put the tools into the well slowly and to withdraw them slowly. Under such conditions it is usually advisable to use the circulator or rotary systems, so that pump pressure can be put on the well at any moment.

It has been the experience in drilling with mud fluid by the use of either rotary or cable tools in California, Texas, and Louisiana, that more than half the instances when wells go "wild" have occurred when the tools were being withdrawn or being run into the hole again.

If there is high-pressure gas in a well being drilled with cable tools, the tools or the bailer should be withdrawn slowly because the rapid withdrawal has much the same effect as swabbing, and may draw in the gas or cause the walls of the hole to cave.

In drilling with rotary tools the drillers may neglect to take account of the displacement of the drill stem when withdrawing it from the hole, so that the level of the fluid may be sufficiently lowered in the hole to cause a blow-out. Trouble has also been caused in cable-tool holes when, to facilitate drilling, the level of the fluid has been lowered too far.

Experience has shown that there is danger in running cable or rotary tools into a hole when the gas pressure nearly equals that of the fluid. When such conditions exist, it is advisable before withdrawing the tools to circulate fresh fluid until all the fluid charged with gas has been removed, and apply pressure with a pump if the gas pressure is high. If these precautions are not taken the well may not blow out when the tools are withdrawn, but may be left in such an unstable condition that it will blow out when the tools are run in again. The agitation of the tools may be sufficient to start the fluid moving, and in a rotary-drilled well the displacement of the drill stem will start the fluid upward and overcome the inertia. The conditions and results are somewhat similar to those in an oil well which may not start flowing until it has been agitated. When such

delicately balanced conditions are liable to exist, it is advisable to put pressure on the fluid with a pump; and in a well being drilled with cable tools and the circulator, that fresh fluid be circulated before putting in the tools. In a rotary well fresh fluid can be circulated after every few stands of drill pipe have been put in. In any case, the tools should be run in slowly and carefully. These principles will be varied in detail according to the particular conditions of each case and the equipment being used. Experience in California has shown that these precautions have reduced the risks of a well going wild.

In territory where the depths to the productive strata are not known, oil or gas sands may be drilled through with mud fluid in the hole and their presence not detected if the driller has not carefully examined the drillings bailed from the hole or the returns if a circulator is being used. This has happened many times. Ordinarily it is preferable not to drill a "wildcat" well with fluid in the hole, but provision should always be made for its use to control any flow of gas that may be encountered.

THE USE OF THE CIRCULATOR WITH STANDARD TOOLS.

The method of using a circulator with standard tools is one developed in recent years, principally in the California fields. The system is essentially the employment of the mud-fluid flushing process of the rotary-drilling system in combination with a string of standard tools. The fluid is forced by a pump through the inner casing and returns between the casing and the walls of the hole, bringing the drillings with it. Sand and coarse materials in the fluid are settled out in a series of settling tanks and troughs, and the mud fluid pumped back into the well. Once drilling has started, there is usually enough mud formed in the process so that no more need be mixed, and at times it is necessary to thin the fluid with clear water.

The extra equipment necessary for using the circulator are a slush pump, hydraulic hose, circulator head, and a system of troughs and tanks for the mud fluid. It is also advisable to have two pumps and an extra boiler, and the rig should be provided with a calf wheel. The same principles of construction are used in the circulator head as in the oil saver. The mud fluid is injected through the hydraulic hose which is attached to the circulator head like the flow lines are to the oil saver.

The use of the mud fluid with the circulator in this manner has certain advantages to which the writers wish to call attention, as well as to precautions which must be taken in its use. It has been especially successful in overcoming loose running sands, bowlders, and caving formations. It has also been successfully used where it

is desired to underream in a hole troubled by running sands or caving formations. By its use the walls of the hole can be plastered more thoroughly with mud fluid than without circulating, the space between the casing and bore hole kept clear of coarse materials, and the pipe kept freer. If the strata are drilled through with mud fluid circulating in the hole the weight of the fluid will hold up the walls of the hole longer, and generally a much greater depth of open hole can be gained than by the use of clear water. The pipe should not be kept too far above the bottom of the hole, because the cuttings will not flush out and will pack around the tools and interfere with drilling. However, under some circumstances it is necessary to drill farther below the casing than in others, and occasionally so far below that the cuttings will have to be bailed out in the usual manner. By this process casing can be carried farther and with less trouble and expense than by other cable-tool drilling systems under the same difficult conditions, and it will seal off the formations more thoroughly and keep the casing freer.



FIGURE 9.—Cross section of well showing how casing may become frozen in drilling by circulator method.

Figure 9 represents a cross section of a well in which mud fluid is being circulated. The cuttings and fluid returning to the surface between the casing and the walls of the bore hole may not be of suffi-

cient volume to fill the space, and consequently the flow will pick out courses on one side of the casing and allow the cuttings to settle and pack on the other side if the pipe is not moved frequently. The passage of the fluid may be free and there may not be any increase in pressure on the circulating pump, but the casing will be found to be tight and in some cases may be impossible to move. Apparently the coarse material settles above the collars and prevents moving the casing upward, but often it will move down of its own weight; or, if not, it can usually be freed by driving a few inches. If the casing can not be freed in this way, the customary methods of moving the casing by jarring, the use of spears, jacks, etc., are employed according to the particular conditions, as though there was no mud fluid in the hole. By moving the pipe as occasion requires, this will be prevented and the fluid will come up uniformly around all sides of the casing and the cuttings will not have a chance to pack. Under some conditions, it is necessary to move the casing as frequently as every 30 minutes. Once circulation has been started it should be kept up continuously until the pipe is landed. Intermittent circulation

frequently results in trouble. Those operators who have understood this principle have been uniformly successful, and wherever it is disregarded trouble is almost sure to result.

When a string of casing is seated in mud fluid in a well drilled by the dry-hole system, the use of a circulator will leave the casing freer and in better condition for withdrawing at any future date than by using mud fluid alone. The fluid is circulated until all coarse materials are flushed from the hole and the walls thoroughly plastered. The casing is then seated and the mud plastered onto the walls, and the weight of the column of mud fluid prevents the shales from slaking and coarse materials from falling into the hole and settling around the collars on the casings.

During the past three years the Keck swinging spider and other similar devices have been used with much success in the Coalinga field, California. This apparatus consists essentially of an especially constructed spider with long reins that reach up over the beam, which works between them, so that it is possible to hold the pipe on the casing lines while drilling proceeds, thus enabling the driller to move the casing at any time he may consider it necessary to keep the pipe free without unhitching or even stopping the motion of the tools. This equipment is of considerable value when it is desired to carry an especially long string of casing through any formation where it is difficult to keep the pipe free. It also allows the driller to regulate conveniently the position of the pipe above the bottom of the hole in order to obtain the most favorable results from circulating. In order that there may be room for moving the casing conveniently it is necessary to have a cellar about 25 feet deep, so that a full joint can be handled.

By means of a circulator, casing has been carried to exceptional depths in the fields where there has been much trouble in carrying casing by ordinary cable-tool drilling methods. In one instance 10-inch casing was carried to 3,336 feet by this means in 122 days' time.* In this way less casing is needed in deep wells, which reduces the expense and permits the operator to finish his well with a larger sized hole. It has been found from experience that in some fields better time is made with the circulator, the well is finished with fewer accidents and in better shape, and at less expense than under other cable-tool drilling methods. Where the conditions are such that much hole must be drilled with cable tools in mud fluid, such as through caving formations, running sands, artesian water or several high-pressure gas sands, it is believed that it would pay the operator to install a circulator.

* Lombardi, M. E., Improved methods of deep drilling in the Coalinga oil field, California: Bull. Am. Inst. Eng., No. 98, 1915, pp. 209-215.

CASING A WELL WITH THE FLUID IN THE HOLE.

Casing a gas well with the fluid in the hole can be accomplished in a few hours without risk to the workmen. On the other hand, several days are often required to case wells that are blowing, and, on account of the danger, \$7 to \$10 a day each has to be paid to men to work in the gas.

Carrying the casing while drilling proceeds has been discussed in previous paragraphs. The use of mud fluid is of much benefit where there are caving strata, running sands, or where the hole must be underreamed. Casing has been carried for many hundreds of feet through such formations, where it would hardly be possible to drill a well without using mud fluid. Properly employed, it is a great aid in keeping the casing free.

If it is desired to place mud fluid behind a string of casing, ordinarily it is not advisable to introduce the fluid from the top. If it is poured in from the top, it carries down all caving and coarse material, and this is likely to lodge against the couplings and freeze the casing, or even to form a bridge which will not let the fluid go past. It is better practice to introduce the fluid through the casing and let it come up under the shoe. In this way cavings and coarse lumps are carried up, and if the fluid is circulated long enough they will be flushed out of the hole, leaving it free from material which may settle or lodge against the couplings. When no means of circulating have been provided, the casing should be raised and lowered a distance of not less than the length of one joint several times before seating. Moving the pipe in this manner tends to grind up cavings and lessens the chances of the pipe sticking and should not be delayed, for if it is not done the casing may be stuck within a few hours after the fluid has been introduced.

Should the casing become "loggy" or "stuck" while mud fluid is being used, the best method of loosening the pipe is to put a steady strain on it rather than try to jar it loose. The conditions may be compared with pulling on a long stick sunk deep into heavy mud: a slow, steady pull will have more effect than a quick, hard jerk. If the pipe has been landed, it may be necessary to let the fluid out from behind it by perforating or shooting the bottom joint.

It is advisable to use heavier pipe when using mud fluid than has been customary in the Mid-Continent and Eastern fields; and at any rate, heavy pipe may be used to advantage on the lower end of the string. It is always advisable to screw the casing together securely.

SEATING THE CASING.

When casing is used to exclude water, oil, or gas, or to retain the mud fluid, it must be properly seated in order to make a tight joint.

A common way of seating casing in the Mid-Continent fields is to land the casing with a thread protector screwed onto the bottom. But this practice does not insure good results, even when sand pumpings have been left in the hole before landing the casing to make a tighter seat, and much damage has been caused by such practices where high-pressure gas or water is found. Often the formation in which the casing is landed is not carefully selected and will not provide a secure seat. The writers know of many instances where failure to provide secure seats has let water or gas out by the packers or under the casing, and much damage has resulted. Instances of this kind are cited on page 32. Not knowing the true source of the water has caused unnecessary expense and loss to many operators.

A better practice, which can often be used to advantage, is to provide a long casing shoe 4 or 5 feet in length. After seating the casing, the hole should be drilled a few feet below the seat and tested for absolute tightness, for a small leak is usually enlarged by either water or gas. Drilling should not be continued unless it has been demonstrated that the casing seat is tight.

A special shoe may be made by taking a "pup" joint or short piece of casing, 6 to 10 feet in length, as conditions may require, of the same size as that about to be seated, screwing on one end an ordinary casing shoe and then slipping on reamed-out couplings until they cover the piece of pipe (fig. 10). The top coupling is then screwed down, so that the collars are all clamped between the top coupling and the shoe. By reducing the last 6 or 8 feet of the hole to the size of the outside diameter of the collars, contact is obtained for the entire length of this shoe with the walls of the hole. If a string of casing has been set with a shoe such as above described, the shoe may cause some trouble in pulling the pipe in the ordinary way, when the well is to be abandoned. However, as the

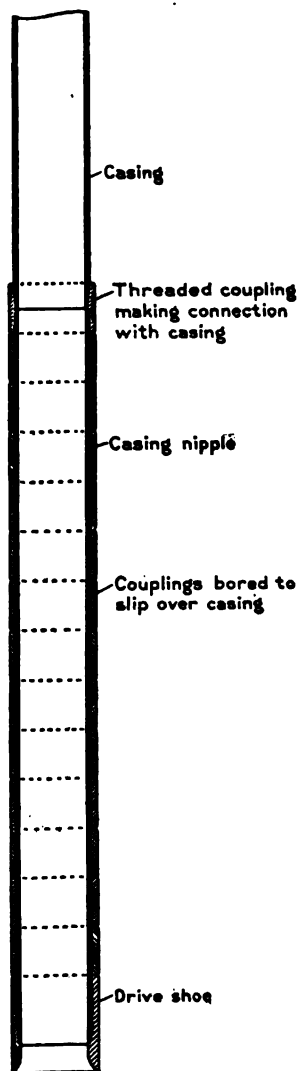


FIGURE 10.—Special long casing shoe.

bottom joint is reinforced with the collars, the string of casing may be started by using a casing spear, which can be used to better advantage than on pipe without collars, as the slips of the spear will not expand the reinforced casing so much.

In some fields, in order to exclude water from the oil-bearing formations, cementing is resorted to. In many cases cementing and trouble from water entering the oil sands can be avoided by using the long shoe.

The mud fluid is less apt to cut under the casing than clear water, because the fluid does not cut the walls of the hole or penetrate small openings so readily as water, as has been demonstrated wherever mud fluid has been used. Some complaint, however, has been made in the Mid-Continent fields about trouble in holding the fluid behind the casing, even when it has been seated with a long casing shoe or with a packer.

Mud fluid subjects the seat to a greater pressure than has been customary in these fields, because ordinarily when the casing is seated there is not much water behind it, although at a later date water accumulates and the increased pressure may break under the casing, leaving the operator in doubt as to whether the water was coming from the producing sand or from some other source. Many instances of this sort are known. The numerous cases in which gas pressure has unseated several strings of casing in a well indicate that the customary practice in the Mid-Continent field does not provide secure seats and is likely to be the cause of damage later in the life of the well.

Another source of trouble was that after the casing had been seated, the fluid was bailed completely from the well and drilling was resumed without any regard to jarring the casing. If the seat is not in a hard formation, the jar of the tools against the casing may loosen it even when a packer is used, and let the fluid escape from behind the pipe. Great care should be exercised in drilling past the seat, and it is advisable to leave enough fluid in the hole to cover the tools in order to lessen the jar against the casing.

Trouble has often followed the use of the following method of placing mud fluid behind the casing.

The casing is filled with the fluid and then raised, letting the fluid come up around the outside. The instant the casing is lifted off the bottom or off the shoulder on which it has been seated, the rock around the bottom of the casing has to stand the sudden pressure of a column of fluid, which in a hole 1,000 feet deep has a pressure of 500 to 600 pounds per square inch when standing. It is obvious that the sudden rush of fluid is likely to dig out the formations around the bottom of the casing so that it will be difficult to seat it again.

THE USE OF PACKERS.

Packers are used in many wells to shut out water or gas, and a judicious use of packers may occasionally save the use of an extra string of casing. These packers are made in various types to suit special purposes, the usual type consisting of two telescoping steel cylinders inclosed in a rubber cylinder, varied in thickness and length to meet different conditions. The component parts are so arranged that when set the rubber expands outwardly and is intended to fill tightly all irregularities in the wall of the well, thus preventing the passage of water or gas. Packers may be used at the bottom of the casing or at some place above.

Whenever one of these packers is removed after being seated it must be repaired or a new packer substituted before a tight joint can be made again.

It has been found from experience that packers frequently develop defects after they have been set, and not much reliance can be placed on them. They will disintegrate under the action of oil or oil vapors, and in many instances large pieces of rubber have blown out of a gas well when the well was being "blown off," showing that packers can not be expected to last indefinitely no matter how carefully they are set. They can not be set successfully in some formations, and the strata at the points where packers should be placed to protect the well properly may not be suitable for setting packers in. The limitations of packers are especially evident where it is desired to confine gas between two packers or where more than one packer is to be used on the same string of casing. No satisfactory way is known to the writers of adequately testing two packers to determine whether both packers have been set without leaks. For these reasons it is believed to be inadvisable to use packers where their failure would result in material damage or where it is not possible to test them at intervals. Where natural conditions do not provide means of testing a packer, mud fluid can be set behind the casing above it, so that if the packer does fail the fluid will run through and either make the leak known or mud off the sand to be protected so that no damage will result.

One of the advantageous features of using mud-laden fluid is that it provides a test for the conditions behind the casing at all times, and the operator can be quite sure as to whether his well is affording proper protection to the oil or gas.

THE USE OF MUD FLUID IN CEMENTING A WELL.

By any one of a number of processes, cement can be forced up outside of the casing, but it seldom extends more than 100 feet above the bottom. The cement reinforces the casing seat and prevents water or gas from cutting under the casing, or in soft, caving formations pre-

vents the pipe following down the hole as the well is drilled deeper. In so far as cementing relates to this report, the points of interest are the effect that mud fluid in the hole will have on the setting of the cement, and to what extent mud fluid can be used to replace the cementing process or as an additional aid in protecting the well. A further point of interest, and one of probable future value, is whether cement, or similar substances, can be used on the same principle as mud fluid and be forced into porous strata, where it will harden and seal off the strata.

The points to be considered in regard to the effect of mud fluid on the setting of cement are these:

Preliminary to the cement being placed, the mud fluid can be made to exclude all oil, gas, or mineralized water from the hole, and as these substances often interfere with the setting of cement, this is a favorable effect. A further advantage is that the mud fluid will aid in keeping up good circulation behind the casing, which is necessary for successfully forcing the cement up around the outside. On the other hand, if mud fluid becomes mixed with the cement to any appreciable extent, it interferes with the setting. The amount of mixing that will take place will depend largely on the process being used. Many operators of wide experience with the cementing process have advocated that the mud fluid be flushed out first and the hole thoroughly washed with water before the cementing process is started, in order to avoid mixing the mud with the cement and to clean the walls of the hole so that a close bond may be formed between the cement and the formations, because it is not likely that the cement alone would scour off all the mud; and any mud between the cement and the walls of the hole will prevent an actual bond.

Other operators object to washing the hole with clear water because it causes caving and interferes with maintaining the circulation and keeping the pipe free. Many operators do not flush the mud fluid out of the hole and yet have had much success in shutting off water when cementing.

When a string of casing has been landed with mud fluid behind it, all water, oil, or gas is excluded from the hole, and if the fluid remains behind the casing the use of cement is not necessary to protect the well. When, however, the formation in which the casing is landed is such that it will not hold a column of fluid behind the casing, or if there is high-pressure gas below the casing which has not been mudded off and may cut away the casing seat, the cement reinforces the casing seat and will be a great aid. Often the same purpose may be accomplished with a long casing shoe. Even if the cement did not make an actual bond with the wall of the hole it would act as a long casing shoe and reinforce the casing seat; or, even if it did not set in an impervious rocklike mass, it would aid in holding the mud

fluid behind the casing. Water might work through the cement or cut a channel between the cement and the walls of the hole, whereas mud fluid will not cut, but will plaster the soft formations and will tend to clog up small openings rather than to pass through them.

When cement only is used, communication between different sands behind the casing will not be prevented, and thus valuable oil and gas deposits may be subject to waste. Neither are the evil effects of unsystematic casing prevented, nor is the casing protected against corrosive waters. In either case much damage may result and make the expense of cementing of no avail, also its presence may make repairing the well very difficult and expensive. If it is possible to cement a well while using mud fluid behind the casing, these evils are prevented, and, furthermore, should any defects develop in the casing or in the seat, the mud fluid will make their presence known and the operator will know the source of his troubles.

The principle of forcing fluid cement or grout into a porous formation to seal it off has been applied with much success in recent years in the Illinois fields, principally by W. W. McDonald, of the Ohio Oil Co., who developed the practical use of the process for shutting out bottom waters in an oil or gas well. The process is described in Bulletin 33 of the State Geological Survey of Illinois.^a A method of shutting off water behind a casing by cementing under pressure, used by C. A. Hiveley, of the Kern Trading & Oil Co., in the Coal-inga field, has also proved successful. Cement has been successfully used in tunnels and shafts to seal off water-bearing sands or creviced rocks. Holes are drilled ahead into the porous rock and fluid cement is forced in under considerable pressure.^b

By this method it is endeavored to force the cement into the sand and hold it there under pressure, so that it will clog the pores of the sand and harden, thus excluding the water from the hole. The essential difference between using cement or other substances which will set and in using mud fluid is that the mud does not harden sufficiently to prevent its being forced out of the sand by pressure, and it is usually necessary to maintain a column of fluid in the hole to keep the sand sealed. When cement or similar material has once set and excluded the water, it is no longer necessary to maintain a counterbalancing pressure in the hole.

The use of cement in this way opens many interesting possibilities, but how far it may be extended and how to apply it under varying conditions can only be worked out by actual practice.

^a Kay, F. H., *Petroleum in Illinois in 1914 and 1915*: Bull. 33, State Geol. Survey, 1916 (advance extract), p. 22.

^b Donaldson, Francis, *The Injection of cement grout into water-bearing fissures*: Trans. Am. Inst. Min. Eng., vol. 48, February, 1914, pp. 136-140.

ADVANTAGES OF CASING A WELL IN MUD-LADEN FLUID.**PROTECTING UPPER OIL OR GAS FORMATIONS.**

It often becomes necessary to case off oil or gas-bearing sands of lesser commercial value in order to reach more valuable sands below (Pl. II, A), and it sometimes happens that water-bearing beds are in close proximity to these sands.

Under the dry-hole method of drilling, extra casing and packers are required to protect fully each one of these formations and as the extra casing delays the work and increases the cost, it is seldom put in. If water occurs below the oil or gas, with no impervious stratum between in which casing can be seated or a packer set, it may be impossible to protect the oil or gas by ordinary methods.

If, however, mud fluid is used, the water will be prevented from injuring the oil or gas. The fluid will drive both the water and the oil or gas back from the hole, and fill the space between the casing and the walls of the hole so that extra casing will be unnecessary. Each porous stratum is sealed off and the contents prevented from entering the hole or moving from one formation into another. (See fig. 11.) In this way, small deposits of oil or gas can be preserved for future use when they may yield greater profits, whereas if they were not protected, they would have yielded the producer little or no returns.

In some fields there are so many different sands or there may be such alternation of water sands with oil and gas sands that it would be impracticable to put in enough strings of casing to protect the valuable deposits unless mud-laden fluid was used. By the judicious use of the mud-laden fluid much casing can be saved and many minor deposits of future value may be protected with slight expense.

MUD FLUID AS A REMEDY FOR UNSYSTEMATIC CASING.

Figures 12 and 13 show the principles of unsystematic casing. If an open space is left behind the casings and they are not landed uniformly according to a system throughout the field in relation to the various sands, a migration of oil, gas, or water will take place that will greatly increase the underground waste of oil and gas, or allow the water to invade oil and gas sands. If there is not some means of getting all the operators in a district to follow the same system in casing wells, great damage may result; as has been proved by experience in the California fields, in the Cushing field, and elsewhere. By placing mud fluid behind the casings such migration and damage can not result.

EFFECT OF MUD FLUID ON KEEPING CASING FREE.

The effect of mud-laden fluid behind the casing, when properly used, is known to be favorable. It has been found from experience



A. TWO ADJOINING WELLS THAT PRODUCE FROM DIFFERENT SANDS.



B. OIL WELL CONNECTED TO FLOW LINE, SHOWING THE TEE CASING HEAD AND FLOW LINE CONNECTED TO SIDE OUTLET.

that it will aid in keeping casing free so that it may be recovered more readily at some later date. The following statement gives

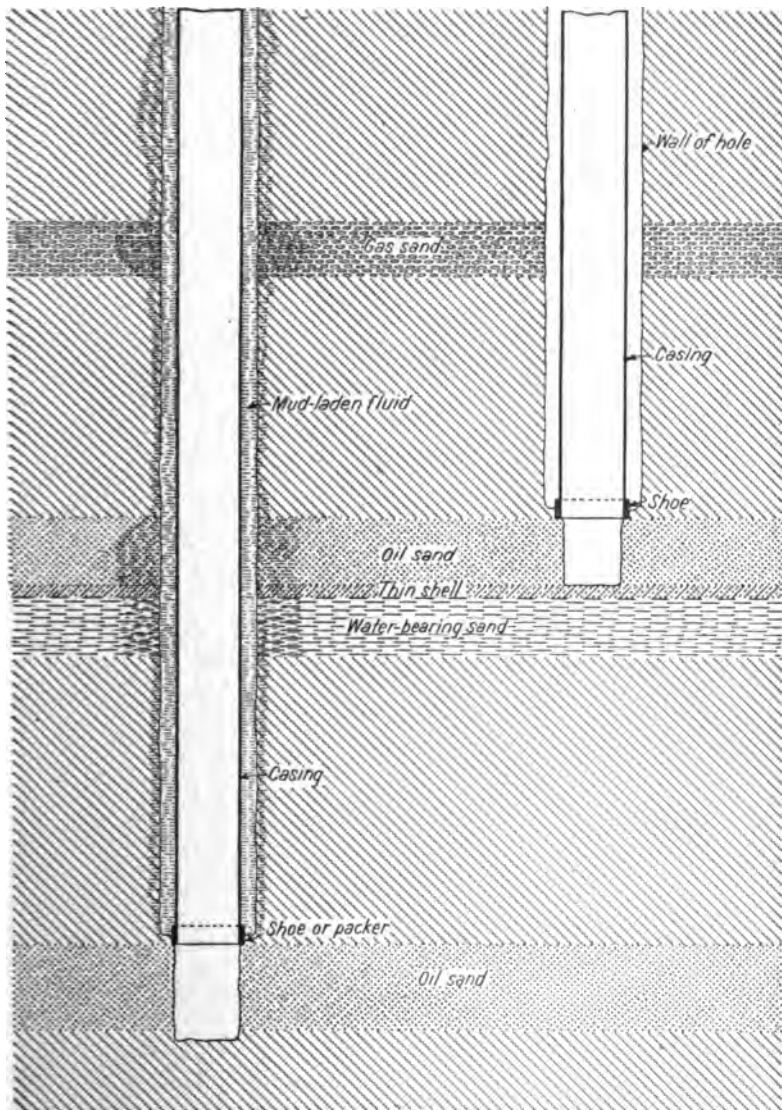


FIGURE 11.—Sealing upper formations by mud-laden fluid method. Well sections are generalized.

the experience of C. P. Clayton, the general superintendent of the Producers Oil Co., in drilling wells in Texas and Louisiana :

Setting casing in thick mud has given successful results without exception. My experience has been limited to the Louisiana and Texas fields, but has extended over a number of years and many varying conditions. Our practice

is to carry our drilling mud thick enough to plaster the walls of the hole and drive the mud into any porous rock sufficient to produce a coating on the inside of the hole impervious to circulating water. Then the casing is set in thick

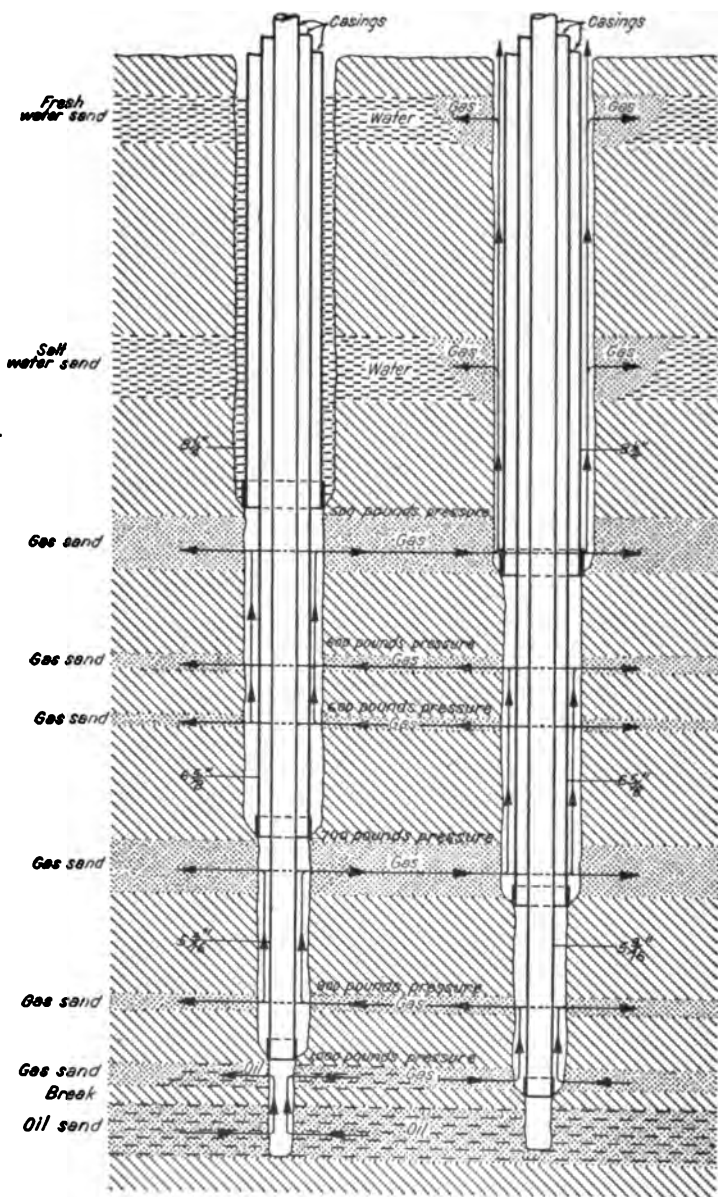


FIGURE 12.—Generalized section showing unsystematic casing of wells. Gas pressure greater than water pressure.

mud, and not only the space between the wall and casing filled with mud, but also the space between the outer and inner casing. We are so particular to insure a perfect filling, with heavy mud and not other material, that it is our

to pack the space between the casing at the top with rope to prevent sand or other substances falling in before the space can be filled with mud.

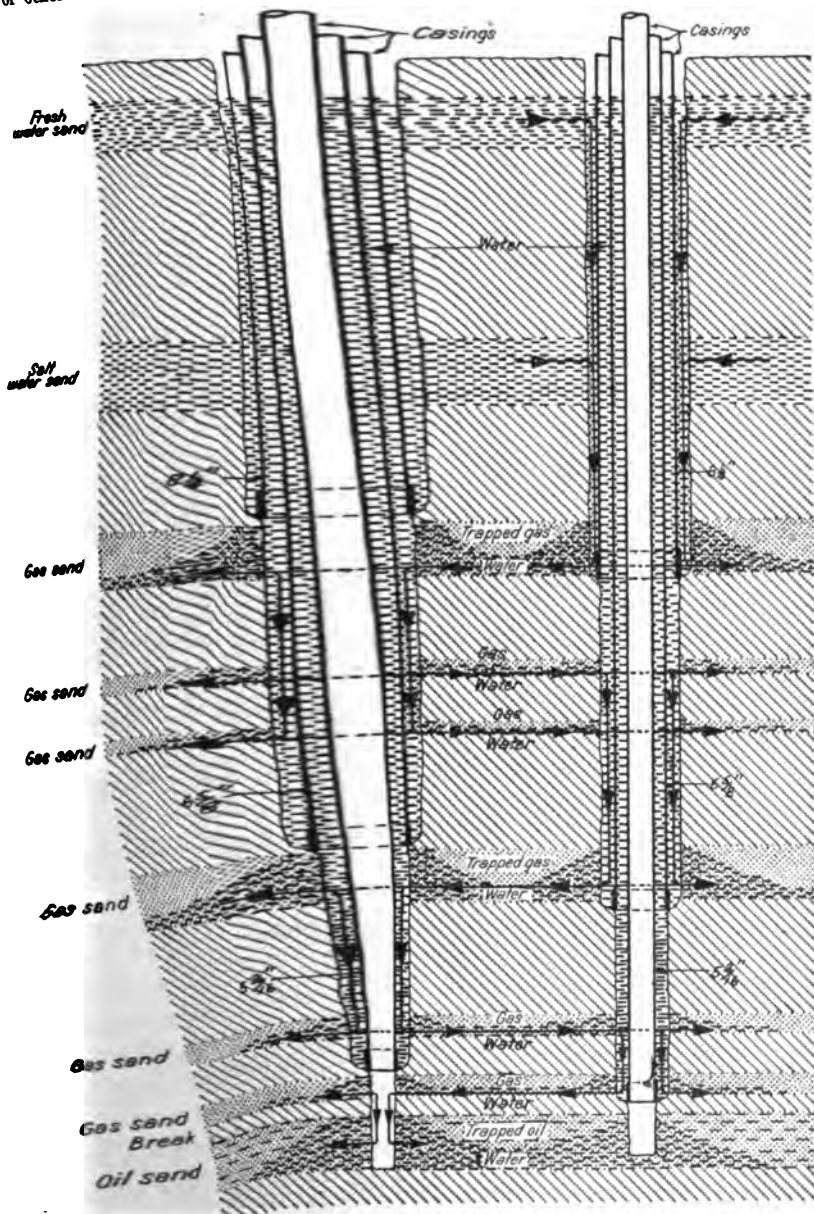


FIGURE 13.—Diagram showing how gas may be wasted by improper casing. The gas is in sufficient volume and under enough pressure to overcome the water; it works up, from the sands of highest pressure, behind the casing and back and forth through the other formations. Much of the gas is lost, and the pressure is greatly reduced. Some of the gas may reach the surface and escape in the air.

You note we make special endeavor to see that the spaces outside of the casing and between the various strings of casing are completely filled with mud.

Mud does not harden sufficiently to prevent pulling the casing. Casing set 5 or 10 years is pulled without difficulty, if properly mudded. Mudding prevents rusting and ordinary disintegration of pipe. Casing set in mud five years was found not to be pitted, whereas casing set in the same field without mud was worthless at the end of five years. Of course, there was some deterioration, but this was offset by the advance in price of casing.

Similar results have been obtained by another large operating company in Texas. In the Lima field, Ohio, drillings from shale were put behind the casing in several hundred wells, but no difficulty was experienced in pulling even second-hand casing. In California casing is so seldom pulled, once it has been set, that there is no definite information on the subject.

Under especially severe conditions of caving or running sands, mud fluid may not permanently hold up the walls and the casing may not be kept free. But even under these adverse conditions the use of mud fluid has been recognized to be more successful than any other method for keeping the pipe free.

MUD-LADEN FLUID PROTECTS CASING FROM CORROSION.

Corrosive waters are a frequent source of trouble and expense in wells. In many fields casing underground is eaten through in two years. The water then becomes a source of much trouble, harming productive strata and eating any other casing or tubing with which it comes in contact. The operator is often misled and may abandon a well, thinking that the water has its source in the productive strata, or he may go to great expense in making ineffective repairs before the true cause is discovered. When a casing has been eaten through, it is often difficult to repair the well, for the casing is likely to be so weakened that it can not be pulled out without being parted; thus the only remedy may be to insert inner strings which in time will also be eaten through, and then the well must be abandoned.

When corrosive waters are present, no ordinary means will fully protect the wells no matter how elaborate and expensive the efforts may be. No combination of casing and packers, nor seating casings with cement, will prevent ultimate damage to the well so long as the destructive water comes into contact with the casing.

Much effort has been made to discover means to protect the casing from corrosive waters. The casing has been coated with tar or other protective substances, but with little favorable effect. Such materials are apt to be worn or scraped off in spots when being placed in the hole. Operators have even gone so far as to use galvanized casing and tubing to prolong its life, or have used casing made by special processes or from alloyed steel. Many claims have been made for the value of the old-style wrought-iron pipe to meet these conditions, but

the fact remains that the wrought-iron pipe now obtainable resists corrosive waters no better than steel pipe. The writers know of no material which has proved satisfactory within reasonable costs.

The writers believe that placing mud-laden fluid behind the casing will effectively protect the casing from water, provided the water is not coming from the inside. The column of fluid exerts greater pressure than the water, drives it back from the hole, clogs up the sand, and prevents the water from entering the hole. It follows as a matter of course that the water does not have access to the casing and can not harm it so long as this condition is maintained.

At Lima, Ohio, several hundred wells of one company have been successfully protected by the use of shale drillings. The manager of this company states that formerly casing of either iron or steel would not last longer than two years when placed underground, whereas casing protected in this manner has lasted for over 10 years with no evidence of corrosion.

Two of the largest operating companies in Texas and Louisiana have found that landing their casing in heavy mud fluid prevents rusting and ordinary disintegration of the pipe. As has been previously stated, casing set in mud fluid for five years was found not to be pitted; whereas, casing in the same field seated without mud fluid behind it was worthless at the end of five years.

In California this favorable effect of mud fluid has not been generally appreciated.

FINISHING A WELL.

Closely related to the subject of the proper method of casing and finishing wells, is that of arranging fittings on the head of gas and oil wells.

THE MASTER GATE.

It is a frequent practice in some fields, when a string of casing has been landed, to cut the last or top joint on a level with the derrick floor, and bell out the top with a sledge hammer to present a smooth surface to the drilling line. It is impossible to put screw fittings on the top of a well with the casing thus battered, and usually the well is allowed to go wild if gas is encountered unexpectedly. The short joint of battered casing must be removed while the well is wild, and a new joint, cut and threaded to proper length, must be put on, after which the usual screw fittings for closing in a well may be applied. The same objections apply to drilling nipples. In all wells where there is any reason to believe that high pressures may be encountered, better means of control should be provided. For a comparatively slight extra cost, a master gate or similar device can be placed on the

head of the well, as shown in figure 14. This master gate should be packed with asbestos packing throughout, so that if the well catches fire the packing will not be rendered useless. It is also recommended that a pipe T wrench 20 feet long be attached to the wheel of the master gate in a lateral position, so that the valve can be operated from a point 20 feet from the hole. In case of a fire or an accident to the fittings above the master gate, the well can be controlled by closing the valve. These comments apply with equal force to the present method of finishing a high-pressure oil well.

Under the ordinary method of shutting in a high-pressure gas well, arrangements of the fittings is such that should the well catch fire from any small leakage around the casing head, or should one

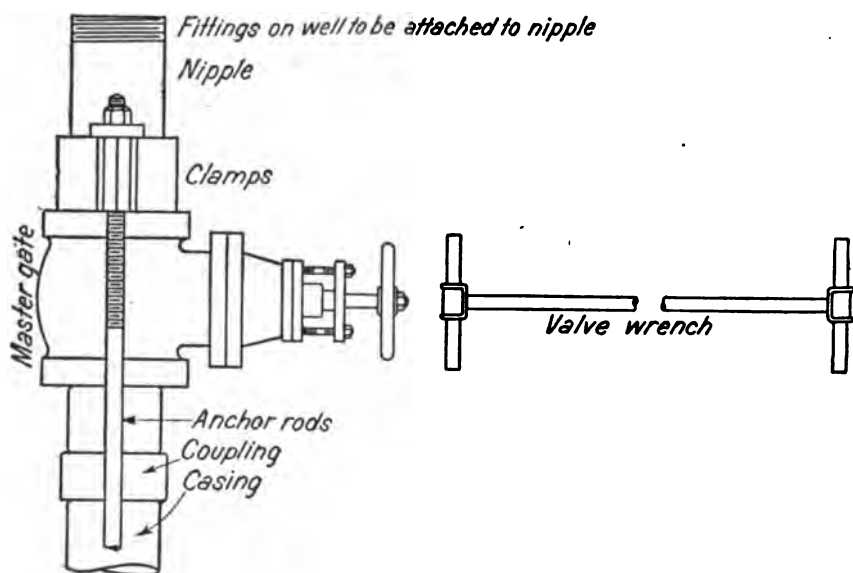


FIGURE 14.—Master valve fitted on high-pressure well.

of the valves become foul, the flow of gas could not be controlled. When a well equipped in this way catches fire, as may frequently happen, the head of the well may have to be broken off by a shot before the fire can be extinguished and the well controlled, and the expense and loss to the operator may amount to thousands of dollars. The purpose of the master valve is to serve as a precautionary measure to provide a means for controlling unexpected or strong flows of oil or gas, or to shut off the well in case of danger from fire, or in other emergencies.

It is the custom in the Mid-Continent fields to equip high-pressure oil wells with a casing head, to which are connected flow lines of various sizes. At many wells as many as four 2-inch flow lines are

connected to one casing head, the cap being held in place simply by four set screws. This arrangement would be shown in figure 15, were the master valve omitted.

Should the well catch fire, such an arrangement is entirely inadequate. In order to get control of the well, thousands of dollars may have to be spent. If, as is shown in figure 15, a master valve be placed below the fittings in the manner recommended for high-pressure gas wells, the well can readily be controlled, making it an inexpensive job to put out the fire and repair the damage.

DANGER FROM INADEQUATE FITTINGS.

In Plate II, *B*, which shows the usual method of connecting up an oil well, the top opening of the tee is closed by the casing top, which sets completely within the tee and is held in place by four sharp-pointed set screws, arranged radially through the top of the tee. The hole in the casing top, originally provided for the tubing to pass through, is closed by a wooden plug driven in from the under side. The device is useless under high pressure and is dangerous under moderate pressures.

Too much emphasis can not be placed on the necessity for using adequate fittings on high-pressure wells. In spite of the advice of one of the bureau's representatives, fittings designed for a working pressure of only 250 pounds were used on a well having much greater gas pressure, with the result that these fittings were immediately blown up when the well was closed in. Sometimes such fittings have been used for higher pressures without accidents, but it is very risky because, although all fittings are designed with a large margin of safety, they can not be used safely beyond their indicated working pressures, and it is not only dangerous to life and property to employ such fittings, but it is false economy.

On all high-pressure wells especial pains should be taken to anchor the fittings in an adequate manner, and reliance should not be placed entirely on threads, especially in cast-iron fittings.

FINISHING A COMBINATION OIL AND GAS WELL.

In some wells an attempt is made to save the gas by shutting it in between two strings of casing which have a stuffing-box casing head, known as a bradenhead, at the top. Usually there is a consider-

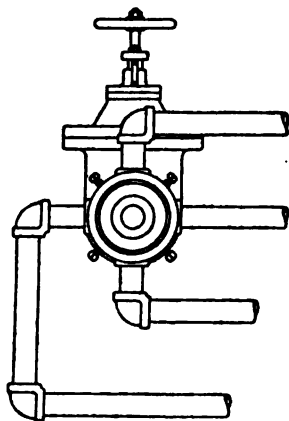


FIGURE 15.—Flow pipes fitted to casing head, with master valve. Top view.

able distance between the bottom of the one casing and the bottom of the other where the gas is confined in direct contact with the strata penetrated, some of which are usually porous (see fig. 1). Much of the gas enters the more permeable strata, sometimes forcing itself to great distances, and is lost, so that when the well is opened at a later date the gas has escaped to such an extent that the yield is of practically no value to the owner. In some instances it has constituted a formidable danger by forming a stray gas sand, which has resulted in fires at gas wells with attendant loss of life and property. Such a fire was caused in the Cushing field.* The oil and gas in a field can not be completely protected so long as they have access to open formations, and this should not be permitted.

Taking gas from between casings through a bradenhead or similar device is not advocated by engineers of the Bureau of Mines, because the gas can not be cared for properly, and it has been found from experience that such a method of producing gas is not satisfactory either in regard to conservation or in the returns to the producer. The pressure and volume are greatly reduced by passing the gas up between the casings, and the flow generally lasts but a short time. If there is open hole either above or below the gas, there is danger of underground waste. If the open hole is above the gas, cavings drop in against the pipe and in time obstruct or entirely prevent the passage of the gas. Even where casings are seated directly above or below the gas, the result is not satisfactory, for there is no way known of ridding the gas of water, and thus the water accumulates and may drown out the gas. It is believed that it will be found a more profitable practice to drill separate wells for the gas. Gas wells drain a far larger area than oil wells, hence it is unnecessary to make each oil well also a producer of gas. A combination well seems justified only as a temporary measure, and after the temporary needs have been served the sands should be mudded off, then much of the casing may be recovered.

In drilling a combination oil and gas well by the mud-laden fluid method, the fluid is put into the well just before the gas sand is reached, after which drilling proceeds to a point just below the gas sand, and the next string of casing is inserted. Before this final string of casing is seated on the bottom, which can be done either with a packer or shoe, as the case may be, the fluid inside the casing is bailed down, that on the outside of the casing being lowered at the same time. A bradenhead is attached to the next outer string of casing and packed. The gland of the bradenhead is prevented from taking a friction hold on the pipe by two or three small blocks of wood. Then, when the fluid has been removed to such depth that its

* See McMurray, W. F., and Lewis, J. O., *Underground wastes in oil and gas fields and methods of prevention*: Tech. Paper 130, Bureau of Mines, 1916, p. 14.

hydrostatic head is less than the gas pressure, the remaining fluid can blow out of the well. The casing is then seated, and the bradenhead bolts, already put in place, are tightened. The seating of the casing in this manner will turn the gas up on the outside of the inner casing and expel through the bradenhead that part of the fluid between the two casings, so that when the well is cleaned, which will not take more than a few minutes, the valves of the bradenhead can be closed and drilling proceed into the oil below in the usual manner. The arrangement of the casing and bradenhead is shown in figure 2. Casings should be landed directly above and below the gas deposit in order to eliminate all danger of underground waste.

A number of methods have been suggested for finishing a well where it is desired to bring gas to the surface between casings without the necessity of landing separate strings of casing directly above and below the gas sand.

Probably some of the suggested combinations of casing and the use of mud fluid, with or without packers, will be successful in recovering the gas without using extra casing and at the same time protecting the gas from other formations. So far as known to the authors, no such methods have been used in actual practice and they are not prepared to say whether the methods will be successful or not.

REOPENING A SEALED STRATUM.

Should it be desired to recover gas from a sand which has been sealed off with mud-laden fluid, the level of the fluid in the hole is lowered until the pressure of the gas is sufficient to force the mud from the sand and to clear the hole. Usually the gas can be recovered in this manner. This has been the experience in the Oklahoma fields, but there are several cases known in California fields, where the sands are looser, that the gas never was recovered in spite of all efforts. If the pressure in a gas sand has been lessened since the mud was forced into it, there may not be sufficient pressure to force out the mud. Washing the hole with clear water may be an aid in recovering the gas, and in a firm sand, such as is found in Oklahoma, shooting may also help.

It should be noted that frequently an oil or gas sand is drowned out by using clear water and the oil or gas can not be recovered, even after the water has been pumped off. It is believed that there is a greater chance of recovering gas that has been sealed off with mud fluid, because the fluid does not penetrate as far into the sand as does water.

It is often possible to unseal oil sands. Many wells are drilled by the rotary or circulator methods through the oil sands with mud fluid in the hole. However, letting mud fluid come in contact with an oil

sand from which it is desired to produce should be avoided when possible, because in general it has an unfavorable effect, especially if the sand is of small capacity or of weak pressure.

USE OF MUD FLUID IN ABANDONING WELLS.

Mud-laden fluid can be used to advantage when a well is to be abandoned, and under some conditions is probably the only practical and economical method of fully protecting the oil, gas, or water-bearing formations penetrated by the well. The usual methods prescribed by State laws of using a combination of wooden plugs, cement, and crushed rock are frequently inadequate and will not properly protect the formations, because there may be such a combination of conditions in the hole that these methods will not prevent waste of gas or the flooding of oil or gas deposits with water. Numerous instances are known where wells plugged as provided by the State law have erupted gas and water, and there are undoubtedly many other instances where damage has gone on which was not indicated from surface evidence. The mud-fluid method is simple and will come nearer fitting all cases than any other method known to the authors.

In using this method the hole is cleaned to the bottom and then is filled to the top with very thick fluid. This drives all oil, gas, or water away from the hole and seals off each formation, so that there can be no waste to the surface or intercommunication between different sands. Care must be taken to use very heavy fluid, so that it will not settle much. In addition to filling the hole with fluid, plugs or other materials may be used where it is thought advisable, but it is considered that the best method is to seal off all strata with the mud fluid before anything else is put into the hole.

This method is especially adapted to conditions similar to those described on pages 57 and 58, where the ordinary system of plugging with wooden plugs and rock would have been of no value and neither packers nor cement could be set. It is also advantageous in wells where casing is to be left in the hole or it is desired to leave the well in such a condition that at a later date it may be opened up and brought to production under changed economic conditions. The productive stratum can be protected from direct contact with the fluid by bridging above the stratum or filling the part of the hole through the stratum with sand or similar material, so that the mud is prevented from entering the productive sand. The methods of protecting the sands will have to be varied according to the conditions in each well.

COST OF USING MUD-LADEN FLUID.

Properly applied, the extra cost of using mud-laden fluid will not be great, and, in fact, can be and has been used in such ways as to

effect an actual saving in the construction of wells. Wherever the use of mud fluid has been advocated, it has been under such conditions that the first cost will be lessened or the extra cost will be more than compensated by a saving of gas and by the favorable condition in which it will leave the well. The principal ways in which the use of mud-laden fluid may be employed to effect savings are:

1. By preventing surface waste of gas.
2. By preventing underground waste of oil and gas and by protecting minor oil and gas sands.
3. By controlling "wild" wells and preventing fires and other accidents.
4. In drilling through high-pressure sands or through caving formations and running sands.
5. By the use of less casing and fewer packers.
6. By protecting the casing against corrosive waters and ordinary disintegration.
7. By shutting off strong flows of water and protecting fresh water.
8. In repairing and abandoning wells.
9. By leaving the wells in such condition that repairs, damages, and production costs will be less during the life of the property, and the value of the property will be generally increased.

The operator is advised to make provision in advance for preventing trouble and waste rather than to stop them after they have started and have done much damage.

The extra cost of "mudding in" a string of casing is often nominal. Operators in Oklahoma have reported to the engineers of the Bureau of Mines that the actual cost of mudding in a string of casing has been as little as \$12, and in a number of instances the average cost was \$30. Sometimes the increased wages alone demanded by casing crews for working around gas will be as much as the cost of using mud fluid. The actual cost will, of course, vary with the conditions and the efficiency of those using the method. The figures quoted were for simple conditions, but if foresight is used by the operator, usually the well can be mudded in before difficult conditions develop and while conditions are still simple.

If there is much caving hole, running sands, high-pressure gas, or strong flows of water, drilling with cable tools in mud fluid may be cheaper and even faster than in a dry hole. Under ordinary conditions, however, the actual drilling costs are less in a dry hole, and if there are none of these difficulties, and no oil or gas to be protected, there is no necessity for drilling in mud fluid. Where the gas is found in commercial quantities the saving in gas alone more than compensates for the time lost in drilling in mud fluid, even when there is nothing to interfere with the speed of drilling in a dry hole.

Most drillers have an erroneous impression in regard to the speed of drilling with mud fluid. It is hardly any slower than drilling in clear water, and in caving or running ground it will be much faster. In the cases where it has been efficiently used in the Cushing field it has been found that it is not any slower than the dry-hole method, because no stops were necessary on account of high-pressure gas and other troubles. The prevention of accidents and fires, to say nothing of the lives saved, in high-pressure gas fields are strong arguments for its use.

By using mud fluid one company completed several wells in record times in a part of the Cushing field, where the high gas pressure had caused much trouble in other wells. Drilling was conducted in mud fluid from the time gas was struck until it had been sealed in and cased off. One well was drilled to a depth of approximately 2,700 feet in 33 days. Not only was the gas saved, trouble and expense avoided, but the company was enabled to obtain the advantage of getting its wells into the Bartlesville oil sand before its competitors.

An important saving, and one readily translated into dollars and cents, is in casing. With the mud-laden fluid less casing need be used in drilling a well, and less need be left in after it is finished. That which is left in is protected against corrosive waters and ordinary disintegration. Loss from corrosive waters does not stop with the damage to one string of casing, but will affect other casings, or tubing, and will increase production costs and injure the productive formations. Defects of this sort are certain to lessen profits and affect the value of the property. Mud fluid not only protects the casing, but should defects develop it will usually indicate the source of the trouble or prevent damage.

The extra items of cost consist of the lubricator (or other devices for introducing the fluid), the pump, preparation of the mud fluid, labor, and time.

If the fluid is introduced before the gas is struck, the lubricator is not necessary and the special slush pump can be dispensed with. The lubricator is constructed of standard fittings, so that afterwards the parts can be used around the property or readily disposed of. The slush pump can be used on many wells, and if it is a small one it can be used for other purposes around the lease. Usually a small company can arrange to rent a pump. It is not justifiable to charge the full cost of the lubricator and pump against one well.

If the operator has saved the drillings from shales and clays and kept them free from sand, etc., it will cost him very little to mix his fluid; but if he has not done this, it may be necessary to haul material for making fluid at considerable expense. If the operator has made provision in advance, the items of labor and time will be very

small. Ordinarily a few hours is sufficient to control a well after the materials and apparatus have been assembled.

By the application of these principles one company has effected an average saving in first cost of \$500 for a large number of wells recently drilled in the Kansas fields, to say nothing of the gas saved and the protection afforded the wells.

Very often the saving in gas is not considered in taking account of the cost of using mud fluid. Numerous instances can be cited from the Oklahoma fields where the gas wasted from wells that could be easily controlled by mud fluid has amounted to many thousands of dollars at current field prices. During a period of 30 days in the spring of 1916 the value of the gas wasted from 12 wells in the Shamrock Pool, Oklahoma, was estimated by representatives of the Bureau of Mines to be \$52,800, at the field price of $2\frac{1}{2}$ cents per 1,000 cubic feet.

By using mud fluid such waste of gas can be prevented, minor oil and gas sands can be protected and made sources of future profit, operating costs can be reduced, and the value of the property increased. The conditions which permit waste of gas ultimately lead to encroachment of water and other evils throughout the field, which sooner or later increase costs, lessen profits, and reduce the amount of oil and gas that can be brought to the surface.

Effective conservation in a district means that the market will be more extensive and more stable. Greater expenses will be warranted in gathering and distributing gas. The marketing companies can afford to pay more for the gas, because the cost of equipment will be apportioned to a far larger quantity of gas, and to enter into more favorable contracts and effect economies which only an assured future supply will justify. These facts will more than offset the fear of some operators that the increased quantity of gas will depreciate its value. By adequately protecting the gas, the producer is in better position for making favorable disposal of it, for he will have a greater quantity to sell and will not have to sell it quickly for fear it will waste away. In recent months the field prices in Oklahoma have been considerably increased, in spite of the great quantities of gas developed and the greater proportion now being saved for use. The producers are finding that they are getting into a better position for bargaining.

A few operators have reported wells where the cost of using mud fluid has been very high. It must be noted that the method has been used under the most difficult circumstances, very often after other methods have failed and left the well in damaged condition, or where not using it was entailing an enormous waste of gas. Under the same conditions of high pressure and large volumes of gas,

there have been frequent cases where wells have cost an excessive amount employing other methods. The operators have not always given the process a fair trial and, in nearly every case, the excessive expense can be traced back to this fact, or that they have not made adequate preparations nor followed directions. A careful reading of the following demonstrations will show these facts. In some cases a large part of the expenses have been due to unnecessary delay where the operator either did not make provision in advance, when he knew that he was going to strike high-pressure gas which should be saved, or he made no effort to expedite the work. Furthermore, some operators, in their estimates of expense, did not take into account the value of gas which would have wasted, nor did they consider the ultimate saving by the protection afforded the well nor make an effort to take advantage of the methods by which casing can be saved.

THE USE OF MUD-LADEN FLUID IN THE MID-CONTINENT FIELDS.

GENERAL CONDITIONS IN THE MID-CONTINENT FIELDS.

The natural conditions affecting the conservation of oil and gas are unusually complex in the Mid-Continent fields. There are a large number of sands containing gas, oil, or water, so that the drill may penetrate many different sands and in every conceivable combination. Figure 16 (p. 52), Plate III (p. 58), and figure 18 (p. 62) show the combination of sands found in parts of the Oklahoma fields. At Blackwell, Okla., gas sands are found at intervals from within a few hundred feet of the surface to a depth of 3,400 feet. Gas is nearly always found associated with an oil pool and may be in the higher parts of the same sand as the oil or in sands above or below the oil sand. Pressures as great as 1,300 pounds per square inch have been found in some of the deeper wells, and flows of over 50,000,000 cubic feet daily have been reported from a number of wells. The formations resemble those in the fields of States farther east, and the wells are usually drilled under contract by the dry-hole method.

Under such complex conditions drilling in a dry hole is wasteful and dangerous, and to fully protect all the formations by the use of casings, packers, and bradenheads is often either impracticable or impossible. Occasionally the gas will interfere or prevent drilling until the pressure has been reduced. The gas is developed faster than it can be marketed, and to finish the well so as to keep it stored in the ground without waste delays the completion of the well by the dry-hole method and adds to the expense.

In the Cushing field the incentive of obtaining unusually prolific oil wells below the gas sands and minor oil sands led operators to disregard the protection of such sands, with the result that probably not more than 10 per cent of all the gas was utilized, and on many properties it was necessary to purchase fuel within a year from the time they were first developed. Furthermore, the inefficient operating methods led to enormous underground waste and to other conditions which have reduced the recovery of oil and added to the expense. Had the mud-fluid methods been used as demonstrated early in the history of the field by the Bureau of Mines, billions of cubic feet of gas would have been saved at a cost far less than its selling value.

Although it was soon demonstrated that the use of mud fluid would prevent waste, there were no adequate State laws to compel the conservation of the gas. The Department of the Interior had the

authority to enforce conservation on the Indian lands, which approximate one-fifth of all the oil lands in Oklahoma, but found it impracticable to prevent waste on most of the

Indian lands so long as waste continued in the rest of the field, for the gas moves so freely throughout the sand that it is almost useless to conserve it at any one well, when wells near by do not protect the gas. For the same reasons, those operators who desired to save the gas were powerless so long as their neighbors would not.

OBJECTIONS TO THE USE OF MUD FLUID IN OKLAHOMA.

Since the mud-fluid method was introduced into the Mid-Continent fields by engineers of the Bureau of Mines in March, 1913, it has been used in over 200 wells whose total open flow was much in excess of a billion cubic feet of gas per day. The cases handled included most of the largest and most difficult wells found during this period. In many instances the method was used after the operators had exhausted all customary means. It has been used to control high pressures, large flows of gas, and wells with defective casing, and to conserve minor sands that could not be protected by other means, because of conditions or the expense of shutting them off.

The method has gone through the period of severe criticism incident to the use of all new or unfamiliar methods. Objections were directed partly against the use of the method itself and partly against the application of conservation to the Oklahoma oil and gas fields. These objections have been largely removed as the operators have become more familiar with the method and its practical utility has been demonstrated. One company alone has "mudded in" over 40 wells in a Kansas field where prevention

of waste is not compulsory, and has not only protected the gas but has found that the first cost of the wells has been considerably re-

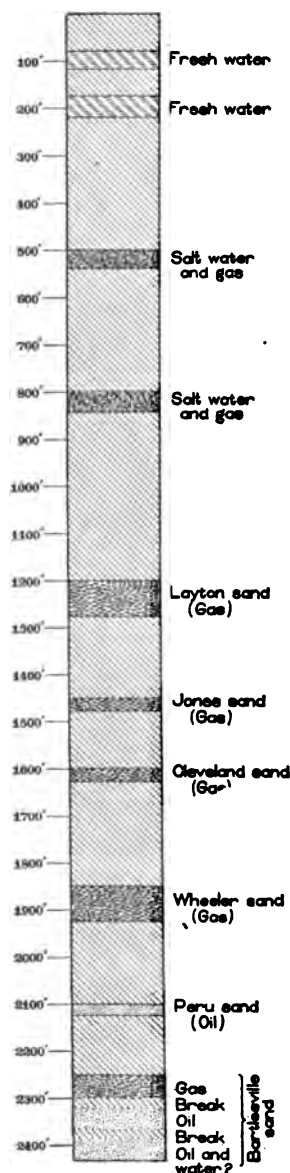


FIGURE 16.—Generalized section showing typical gas and oil sands in north end of Cushing field.

duced. In practically every case the troubles reported can be traced back to not using the method efficiently. Prejudice, unfamiliarity, and not making adequate preparations in advance, have been the principal causes of difficulties.

The nature of the objections to the use of the mud-fluid method in Oklahoma may be gained from those brought forth at a hearing before the Corporation Commission of Oklahoma in the latter part of October, 1915. This hearing was called at the request of certain producers in regard to the order promulgated by the commission for the enforcement of the recently enacted gas-conservation law. Some of the largest producing companies in the Oklahoma fields were represented at this hearing, particularly those operating in the Cushing field. It is to be presumed that the objections and arguments presented at this meeting, represented the best opinions of those objecting to the use of the mud-laden fluid at that time. It was the general sense of those present that they were in hearty agreement with the purpose of the law to conserve gas, but that the order of the corporation commission was impracticable, put the producer under an unnecessary expense, and that the facts of underground waste in their opinion did not warrant some of the prescribed measures.

The points brought out by the objectors at this meeting were:

1. Denial of the importance of underground waste and of the necessity for protecting the gas from open hole;
2. That drilling could not be conducted in mud fluid;
3. That drillings could not be bailed from a hole filled with mud fluid;
4. That the mud fluid would settle and dry out behind the casings;
5. That practically all formations in Oklahoma contained sand, and are not suitable for making mud fluid;
6. That mud fluid would cause the tools and the casings to stick.
7. That its use would cause trouble in drilling and additional expense.

A careful reading of this bulletin and the publications referred to in it will supply the answers to these objections.

DEMONSTRATIONS AT WELLS IN OKLAHOMA.

In order to illustrate the use of mud-laden fluid in fields like those of Oklahoma, the following cases are cited. They have been selected to show the diversified conditions under which it can be used. The demonstrations include the control of difficult wells, the saving of gas, the protection of minor oil and gas sands, the repairing of wells, and other uses. Only cases where representatives of the Bureau of Mines were present are included. Some of the demon-

able distance between the bottom of the one casing and the bottom of the other where the gas is confined in direct contact with the strata penetrated, some of which are usually porous (see fig. 1). Much of the gas enters the more permeable strata, sometimes forcing itself to great distances, and is lost, so that when the well is opened at a later date the gas has escaped to such an extent that the yield is of practically no value to the owner. In some instances it has constituted a formidable danger by forming a stray gas sand, which has resulted in fires at gas wells with attendant loss of life and property. Such a fire was caused in the Cushing field.* The oil and gas in a field can not be completely protected so long as they have access to open formations, and this should not be permitted.

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4. In drilling through high-pressure sands or through caving formations and running sands.
5. By the use of less casing and fewer packers.
6. By protecting the casing against corrosive waters and ordinary disintegration.
7. By shutting off strong flows of water and protecting fresh water.
8. In repairing and abandoning wells.
9. By leaving the wells in such condition that repairs, damages, and production costs will be less during the life of the property, and the value of the property will be generally increased.

The operator is advised to make provision in advance for preventing trouble and waste rather than to stop them after they have started and have done much damage.

The extra cost of "mudding in" a string of casing is often nominal. Operators in Oklahoma have reported to the engineers of the Bureau of Mines that the actual cost of mudding in a string of casing has been as little as \$12, and in a number of instances the average cost was \$30. Sometimes the increased wages alone demanded by casing crews for working around gas will be as much as the cost of using mud fluid. The actual cost will, of course, vary with the conditions and the efficiency of those using the method. The figures quoted were for simple conditions, but if foresight is used by the operator, usually the well can be mudded in before difficult conditions develop and while conditions are still simple.

If there is much caving hole, running sands, high-pressure gas, or strong flows of water, drilling with cable tools in mud fluid may be cheaper and even faster than in a dry hole. Under ordinary conditions, however, the actual drilling costs are less in a dry hole, and if there are none of these difficulties, and no oil or gas to be protected, there is no necessity for drilling in mud fluid. Where the gas is found in commercial quantities the saving in gas alone more than compensates for the time lost in drilling in mud fluid, even when there is nothing to interfere with the speed of drilling in a dry hole.

Most drillers have an erroneous impression in regard to the speed of drilling with mud fluid. It is hardly any slower than drilling in clear water, and in caving or running ground it will be much faster. In the cases where it has been efficiently used in the Cushing field it has been found that it is not any slower than the dry-hole method, because no stops were necessary on account of high-pressure gas and other troubles. The prevention of accidents and fires, to say nothing of the lives saved, in high-pressure gas fields are strong arguments for its use.

By using mud fluid one company completed several wells in record times in a part of the Cushing field, where the high gas pressure had caused much trouble in other wells. Drilling was conducted in mud fluid from the time gas was struck until it had been sealed in and cased off. One well was drilled to a depth of approximately 2,700 feet in 33 days. Not only was the gas saved, trouble and expense avoided, but the company was enabled to obtain the advantage of getting its wells into the Bartlesville oil sand before its competitors.

An important saving, and one readily translated into dollars and cents, is in casing. With the mud-laden fluid less casing need be used in drilling a well, and less need be left in after it is finished. That which is left in is protected against corrosive waters and ordinary disintegration. Loss from corrosive waters does not stop with the damage to one string of casing, but will affect other casings, or tubing, and will increase production costs and injure the productive formations. Defects of this sort are certain to lessen profits and affect the value of the property. Mud fluid not only protects the casing, but should defects develop it will usually indicate the source of the trouble or prevent damage.

The extra items of cost consist of the lubricator (or other devices for introducing the fluid), the pump, preparation of the mud fluid, labor, and time.

If the fluid is introduced before the gas is struck, the lubricator is not necessary and the special slush pump can be dispensed with. The lubricator is constructed of standard fittings, so that afterwards the parts can be used around the property or readily disposed of. The slush pump can be used on many wells, and if it is a small one it can be used for other purposes around the lease. Usually a small company can arrange to rent a pump. It is not justifiable to charge the full cost of the lubricator and pump against one well.

If the operator has saved the drillings from shales and clays and kept them free from sand, etc., it will cost him very little to mix his fluid; but if he has not done this, it may be necessary to haul material for making fluid at considerable expense. If the operator has made provision in advance, the items of labor and time will be very

small. Ordinarily a few hours is sufficient to control a well after the materials and apparatus have been assembled.

By the application of these principles one company has effected an average saving in first cost of \$500 for a large number of wells recently drilled in the Kansas fields, to say nothing of the gas saved and the protection afforded the wells.

Very often the saving in gas is not considered in taking account of the cost of using mud fluid. Numerous instances can be cited from the Oklahoma fields where the gas wasted from wells that could be easily controlled by mud fluid has amounted to many thousands of dollars at current field prices. During a period of 30 days in the spring of 1916 the value of the gas wasted from 12 wells in the Shamrock Pool, Oklahoma, was estimated by representatives of the Bureau of Mines to be \$52,800, at the field price of $2\frac{1}{2}$ cents per 1,000 cubic feet.

By using mud fluid such waste of gas can be prevented, minor oil and gas sands can be protected and made sources of future profit, operating costs can be reduced, and the value of the property increased. The conditions which permit waste of gas ultimately lead to encroachment of water and other evils throughout the field, which sooner or later increase costs, lessen profits, and reduce the amount of oil and gas that can be brought to the surface.

Effective conservation in a district means that the market will be more extensive and more stable. Greater expenses will be warranted in gathering and distributing gas. The marketing companies can afford to pay more for the gas, because the cost of equipment will be apportioned to a far larger quantity of gas, and to enter into more favorable contracts and effect economies which only an assured future supply will justify. These facts will more than offset the fear of some operators that the increased quantity of gas will depreciate its value. By adequately protecting the gas, the producer is in better position for making favorable disposal of it, for he will have a greater quantity to sell and will not have to sell it quickly for fear it will waste away. In recent months the field prices in Oklahoma have been considerably increased, in spite of the great quantities of gas developed and the greater proportion now being saved for use. The producers are finding that they are getting into a better position for bargaining.

A few operators have reported wells where the cost of using mud fluid has been very high. It must be noted that the method has been used under the most difficult circumstances, very often after other methods have failed and left the well in damaged condition, or where not using it was entailing an enormous waste of gas. Under the same conditions of high pressure and large volumes of gas,

there have been frequent cases where wells have cost an excessive amount employing other methods. The operators have not always given the process a fair trial and, in nearly every case, the excessive expense can be traced back to this fact, or that they have not made adequate preparations nor followed directions. A careful reading of the following demonstrations will show these facts. In some cases a large part of the expenses have been due to unnecessary delay where the operator either did not make provision in advance, when he knew that he was going to strike high-pressure gas which should be saved, or he made no effort to expedite the work. Furthermore, some operators, in their estimates of expense, did not take into account the value of gas which would have wasted, nor did they consider the ultimate saving by the protection afforded the well nor make an effort to take advantage of the methods by which casing can be saved.

THE USE OF MUD-LADEN FLUID IN THE MID-CONTINENT FIELDS.

GENERAL CONDITIONS IN THE MID-CONTINENT FIELDS.

The natural conditions affecting the conservation of oil and gas are unusually complex in the Mid-Continent fields. There are a large number of sands containing gas, oil, or water, so that the drill may penetrate many different sands and in every conceivable combination. Figure 16 (p. 52), Plate III (p. 58), and figure 18 (p. 62) show the combination of sands found in parts of the Oklahoma fields. At Blackwell, Okla., gas sands are found at intervals from within a few hundred feet of the surface to a depth of 3,400 feet. Gas is nearly always found associated with an oil pool and may be in the higher parts of the same sand as the oil or in sands above or below the oil sand. Pressures as great as 1,300 pounds per square inch have been found in some of the deeper wells, and flows of over 50,000,000 cubic feet daily have been reported from a number of wells. The formations resemble those in the fields of States farther east, and the wells are usually drilled under contract by the dry-hole method.

Under such complex conditions drilling in a dry hole is wasteful and dangerous, and to fully protect all the formations by the use of casings, packers, and bradenheads is often either impracticable or impossible. Occasionally the gas will interfere or prevent drilling until the pressure has been reduced. The gas is developed faster than it can be marketed, and to finish the well so as to keep it stored in the ground without waste delays the completion of the well by the dry-hole method and adds to the expense.

In the Cushing field the incentive of obtaining unusually prolific oil wells below the gas sands and minor oil sands led operators to disregard the protection of such sands, with the result that probably not more than 10 per cent of all the gas was utilized, and on many properties it was necessary to purchase fuel within a year from the time they were first developed. Furthermore, the inefficient operating methods led to enormous underground waste and to other conditions which have reduced the recovery of oil and added to the expense. Had the mud-fluid methods been used as demonstrated early in the history of the field by the Bureau of Mines, billions of cubic feet of gas would have been saved at a cost far less than its selling value.

Although it was soon demonstrated that the use of mud fluid would prevent waste, there were no adequate State laws to compel the conservation of the gas. The Department of the Interior had the

flow of gas was reported at 22,000,000 cubic feet daily. The total amount of gas wasted is estimated to be not less than 6,000,000,000 cubic feet. The conditions as found in this field are similar to those in the Cushing field and it is evident that no combination of casing or packers would have protected all the gas sands penetrated by the well.

SHUTTING GAS AND WATER OUT OF AN OIL WELL.

The Foster & Davis No. 2 well was located 2,000 feet distant from the White & Sinclair well, in sec. 36, T. 24 N., R. 8 E., and was the only other well in this pool which had been drilled into the Bartlesville sand. It was originally drilled to 2,225 feet and reached the Mississippi limestone. It was later plugged at 2,159 feet to exclude bottom water. The well was completed January 2, 1915, and showed an initial production of 50 barrels of oil from the Bartlesville sand and a large volume of gas. When inspected in August, 1915, it was producing a small amount of "roily" oil and much water; two or three million cubic feet of gas was escaping through the bradenhead. The company believed that the water was coming from the Mississippi formation, which had been plugged off, and that it would be necessary to abandon the well. Instructions were given to shut in the gas but the company claimed this was an impossibility. On the suggestion of Mr. Aggers, mud fluid was introduced by the lubricator method without difficulty behind the 6½-inch casing, which was landed above the Bartlesville sand. The well made 150 barrels of good oil within the next 24 hours and afterwards settled down to a production of 12 barrels of clear oil and 10 barrels of water daily. The use of mud fluid in this case shut off all the gas and most of the water and saved the well to the owners, who otherwise would have abandoned it. It is evident that the 6½-inch casing was not seated securely, as thought by the owners, and that most of the water was coming from above the Bartlesville sand and not from the Mississippi limestone below.

In these last two demonstrations the conditions of the wells made it necessary to introduce the mud fluid between the casings from the top, which as a rule is not advisable, because of reasons given elsewhere in this paper.

REPAIRING A SMALL GAS WELL.

A case which would have entailed considerable expense and trouble by other methods was easily remedied by the use of the mud-laden fluid.

On November 8, 1914, the Indian Territory Illuminating Oil Co. completed a small gas well in sec. 22, T. 25 N., R. 11 E., in the Osage

Reservation. About August, 1915, the well commenced to show salt water, which increased in quantity and did much damage to the surface ground about the well. The tubing was pulled and found to be eaten through in several places by the water. In attempting to repair the well a packer and several joints of tubing were inadvertently left in the hole.

The company was preparing to abandon the well, but upon the advice of Inspector H. D. Aggers the following method was used to repair it. Tubing, with a swedge nipple on the bottom and a packer placed above, was put in the hole and the packer set below the 6½-inch casing, which was landed above the gas sand. The swedge nipple was set over the tubing which had not been fished out of the hole. This shut off the water. When the well was closed in overnight the water, owing to accumulated pressure, burst past the packer. The 3-inch tubing which had been used was then replaced by 2-inch tubing, as had been advised by Mr. Aggers at first, as in his opinion the volume of the gas was too small to lift the water out of the larger-sized tubing. When the packer was set again, it was placed in the 6½-inch casing and all the water was shut off, showing that it was not coming from the gas sand nor from under the casing seat, but had eaten through both the casing and the tubing. Thick mud was then placed behind the casing and between the casing and the tubing. The mud fluid kept the water from entering the well and corroding the tubing. By this method the gas was left dry and in good condition, and the life of a well which would otherwise have been abandoned has been prolonged.

CONTROLLING A HIGH-PRESSURE GAS WELL.

A case similar to the White & Sinclair well No. 2, in the Osage Reservation, was that of the McMan Oil Co.'s well No. 1, on the Mikey farm, located in the southern extension of the Cushing field. The conditions are shown in figure 18.

No attempt was made to shut in the gas until the 8½-inch casing had been landed below the Layton sand, when the Layton gas was shut in between the 8½ and 10 inch casings with a bradenhead. The pressure of the gas unseated the 10-inch casing, thus letting the water, which had access to the Bruner sand, also have access to the Layton sand, affecting a total daily flow of 40,000,000 cubic feet of gas. An effort was made to drown out the gas by letting the water stand on the Layton and Bruner sands. A gate was placed on the 10-inch pipe, but when the gate was closed it was torn off almost immediately, along with a 10-inch nipple, and wrecked two sides of the derrick. At the same time the 10-inch casing was unseated. The gas also cut under the packer on the bottom of the

8½-inch casing. The gas blew the water out of the hole in large volumes, and also reached the surface through many crevices in the ground near the well. Drilling was continued to a depth of 1,975 feet, three more streaks of gas being found, with an additional volume of 5,000,000 cubic feet daily. At this time the company was notified by the corporation commission to repair the well in accordance with the State laws.

On October 1 the field superintendent stated that he had exhausted every effort to repair the well and turned it over to the State conservation officers, James F. York and L. N. McGinley, who requested the assistance of Inspector G. W. McPherson, of the Bureau of Mines. The company was given instructions to provide the equipment and materials necessary for mudding in the well. On October 4 the company furnished a pump, which was, however, of an obsolete pattern, greatly worn, and with valves unsuited for the work at hand. Notwithstanding, an effort was made to mud in the well, in hopes of avoiding further waste. This effort was unsuccessful because the pump could not be run continuously on account of frequent stops for repairs, and furthermore it could not handle a sufficient volume of fluid to overcome the gas. It took the company from October 6 to October 17 to obtain the proper equipment and to haul clay to the well. Lubrication was started at 2 p. m. on October 17, and the well was under control at 5.30 p. m., 500 barrels of fluid being used. Circulation was continued throughout the night

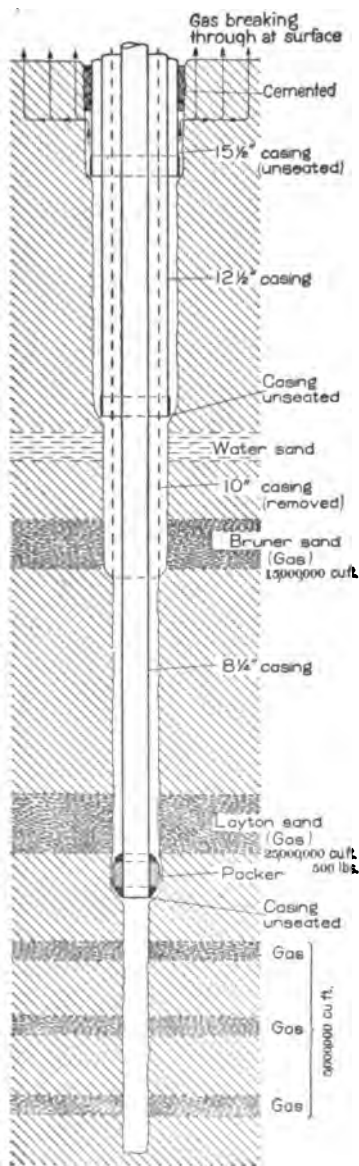


FIGURE 18.—Section showing condition of well No. 1, Mikey farm, south extension of Cushing pool. Formations from driller's record.

and pressure maintained against the Bruner sand by connecting the pump with the bradenhead.

Before this well was killed it was freely predicted by nearly every oil man who had seen the condition of the well that it would be impossible to control, as the mud fluid would be blown out of the crevices as soon as it was put into the well. Lubrication had continued only a short time when the gas and water, which had been erupting in considerable volume at different points around the well, died down and thereafter remained quiet. The reason why the mud fluid did not blow out of the crevices was that the crevices extended down only to a comparatively shallow sand fed by gas derived from the Layton and Bruner sands which came up behind the 8½-inch casing, the 10-inch, 12½-inch, and 15½-inch casing having been removed, and as soon as the gas in these sands had been sealed in there was no longer communication with the sand from which the crevices were being supplied.

After the well had been mudded in, it subsequently blew out while being drilled to the Wheeler sand at a greater depth, because the drillers bailed down the fluid to facilitate drilling and thus reduced its pressure to less than that exerted by the gas. As soon as the well was "lubricated" again the gas was killed and confined to the sands.

From the time the Bruner sand was struck until the well was mudded in the waste of gas is estimated to have been 15,000,000 to 41,000,000 cubic feet per day. Had the company owning this well provided the necessary equipment in advance, as specified in the order of the corporation commission, this waste need not have occurred and the expense occasioned by shutting in the well would have been avoided. The supposed extra cost of drilling in mud-laden fluid to protect the gas would have been much less than that of bringing the well under control, and the gas saved would have repaid many times the additional expense. The attempted use of bradenheads in this case proved to be utterly impracticable and resulted in conditions that caused great waste, and if they had not been corrected by the application of mud fluid would have done irreparable damage to the field. It was demonstrated later on a property near by that under such conditions wells could be drilled faster and cheaper by using mud fluid than by trying to drill in a dry hole because accidents and delays caused by the gas were avoided.

PROTECTING MINOR OIL AND GAS SANDS.

An illustration of how minor deposits of oil or gas may be protected without extra casing and with little expense is furnished by a well located near Sapulpa, Okla. In this well the Perryman sand at 1,296 feet showed some oil and gas, the Red Fork sand at 1,518 feet showed some oil, the Glenn sand at 1,618 feet showed 4,000,000 cubic feet of gas and 50 barrels of oil at the top and a great quantity of

water in the bottom. On account of the water, which could not be separated from the oil sand, the well could not be pumped profitably, and it was therefore deepened to the Lost City sand at 1,903 feet, from which 12 barrels a day was obtained. Then 6½-inch casing was landed below the Glenn sand at 1,793 feet. The 8¼-inch casing was pulled, allowing communication between a water sand at 430 feet and the Perryman, Red Fork, and Glenn sands. For three months the gas from the Glenn sand blew gas and water from the outside of the 6½-inch casing. The pressure then became so depleted that the water overcame it and drowned out the gas. The well remained in this condition for a year, although the sands affected were of much commercial value in that district.

As the well was on restricted Indian land, the company was notified by Inspector J. C. Fowler that it could not be allowed to remain in such condition. At Fowler's suggestion, the space outside of the 6½-inch casing was filled with 75 barrels of mud fluid, at small expense, and to the satisfaction of the owners of the well. It was not even necessary to stop pumping the well while the fluid was being placed behind the casing.

As there was no break between the water and oil in the Glenn sand, it had been found impossible to protect the oil and gas from the water. By using the fluid, no extra casing or packers were necessary, and not only the Glenn sand but also the other two sands were adequately protected. The fluid has been used in other wells where it was desired to drill through an upper productive stratum, in the lower part of which water was found, and possibly its use is the only way in which such deposits can be adequately protected at reasonable expense.

THE USE OF MUD FLUID IN THE ADA FIELD.

The value of having the operations of a field carefully supervised is well illustrated by the work of J. C. Lowrey, in the Ada gas field, Oklahoma. The Ada field is not large, nor does it present unusually difficult problems, but it is of much local importance because it is the only supply of gas in that district and is of great benefit to the town and to the large cement factory located there. Consequently the citizens are desirous of saving the gas and prolonging the supply as far as possible. As usual, the field is divided among many operators and trouble had been experienced in getting them to cooperate. At the request of the Ada Chamber of Commerce, Special Inspector Lowrey was placed in the Ada field in July, 1915, it being arranged that Lowrey's salary and expenses were to be paid by the Ada Chamber of Commerce, and his work was to be under the supervision of the Bureau of Mines. Arrangements were also made for cooperation with the State conservation officers.

After a careful study extending over several weeks, it was found that some of the small number of wells in the field were not in satisfactory condition. Tests were made on these wells, and it was found that water which the operators had thought was coming from the producing sands, was coming under the casing from the upper sands. It was also found that gas sands of minor commercial value were not being protected and that there was danger from unsystematic casing and improperly plugged wells. On Inspector Lowrey's recommendations and through the authority of the State corporation commission and the Department of the Interior these conditions were corrected. Mud fluid was placed behind the casings with much advantage, shutting off all top waters and protecting the minor gas sands.

A peculiar case in the Ada field was reported by Inspector Lowrey. A gas well had just been completed and shut in when it was struck by lightning and the rig burned down, letting the tools which had been hanging in the derrick drop into the space between the casing and the walls of the hole. Gas showing a flow of 3,000,000 cubic feet daily had been "bradenheaded" in between the $6\frac{1}{2}$ and $8\frac{1}{4}$ inch casings. In some manner the tools started a leak in the $8\frac{1}{4}$ -inch casing, so that the water came through and was blown out from between the $8\frac{1}{4}$ and $6\frac{1}{2}$ inch casings. It was not known whether the packer at the bottom had been jarred loose, a leak started around a collar, or a hole cut in the casing. In any event the tools could not be fished out, so that it was unlikely that the casing could be pulled. It was thought that possibly the leak was so small that mud fluid would clog it up and prevent the movement of gas or water through it. Fluid was pumped in between the $6\frac{1}{2}$ and $8\frac{1}{4}$ inch casings from the top and circulated outside the $8\frac{1}{4}$ -inch casing to the surface. Further escape of gas was stopped and water prevented from working into the well, thus leaving the well in good condition.

Another instructive case in the Ada field is that of Allen No. 2 well of the MacThwaite Oil & Gas Co. The well was shut in as a gas well and for a number of days afterwards showed a pressure of 240 pounds, but because there was 34 feet of open hole "bradenheaded" in with the gas sand, Inspector Lowrey advised the company to remedy the condition to avoid possibility of waste. The well was mudded in and the casing reset with a packer at the top of the sand. Within 48 hours the well registered 375 pounds pressure, an increase of 135 pounds over the previous measurement.

REPAIRING A DEFECTIVE GAS WELL NEAR HENRYETTA.

In the course of the inspection of the leases of restricted Indians, Inspector L. W. Courtney found three wells in the Henryetta district which were blowing gas outside of the casing. These wells had

been drilled for three years, but had been shut in and not utilized because they were located some distance from any gas lines. The owners of the wells had believed that the gas was from a shallow sand, which at the time the wells were drilled was not considered of commercial value. In consideration of the fact that gas had been escaping steadily from these wells for three years, Courtney thought that the gas should have been exhausted long before if such had been the case. This matter was taken up with the owners of the wells, who decided to place mud fluid outside the casing. This was done, but in a few days the fluid disappeared and the gas came up outside the casing again. On opening the well, fluid was blown out with the gas, as well as pieces of rubber from the packer, showing that the packer leaked. The casing could not be pulled, so fluid was put inside and outside of the casing. When an attempt was made to pull the casing, it parted and was found to be badly corroded. An inner string of casing was landed with a packer and the fluid bailed from the inside, the gas was recovered, and the well was put into good condition.

This case illustrates how frequently the source of trouble from escaping gas or infiltrating water is from leaky packers or casing, whereas the operator may entirely misinterpret the meaning and the source of the trouble. If mud fluid had been placed behind the casings at first, this defect would probably not have developed; or if defects developed in the packer or casing, the column of fluid would either have prevented damage or by leaking away would have made it evident what the defect was, so that the operator would have known how to repair his well and would have been saved much expense and prevented damage to the field.

LAWS AND REGULATIONS RELATING TO OPERATION OF OIL AND GAS LANDS IN OKLAHOMA.

The technological manner of operating oil and gas lands controlled by the Federal Government is under the supervision of the Bureau of Mines. Because of the mobility of oil, gas, and water, these resources in Government lands can not be fully protected unless effective methods are similarly applied to adjoining territory. For these reasons, the Bureau of Mines is directly concerned in the laws of those States in which Government reserves are situated.

From the experience gained by the representatives of the Bureau of Mines in Oklahoma, it is believed that the following State laws, and the Federal regulations governing operations on Indian lands; include the technological features advisable for the proper conservation of oil and gas. Under the regulations for operating Indian lands, the United States Government is interested not only in preventing waste but in protecting the rights of the Indian wards as lessors.

A careful reading of the facts and references given elsewhere in this paper supplies the reasons why these technological features are deemed advisable. Notwithstanding the well-known difficulties involved in regulating oil and gas operations, it is believed that the recent experiences in Oklahoma show that such laws and regulations can be made and enforced and will largely eliminate waste; whereas a review of the history of the oil and gas industry from its inception to the present time shows conclusively that when left to themselves the operators have not conserved and will not conserve the gas nor even fully protect the oil resources.

For the practical conservation of gas, it is necessary to protect the equities of every lessee or owner of gas lands by providing a method for prorating the gas. It has been the constant complaint of oil men and small producers of gas that, not owning pipe lines, they were unable to market the gas except through the large gas-selling companies, and that sometimes these gas-marketing companies would drain a whole field from a few of their own wells which were favorably located and refuse to buy gas from other producers. If conservation were to be applied without providing for equal access to the market, it would be possible for gas companies to reap all the benefits and allow the producers none. It is very much to the interests of the gas companies to protect their investments and the future markets by conserving the gas, and ordinarily they are anxious to

treat other producers of gas fairly, because unfair treatment would doubtless make the law impracticable of enforcement or cause its repeal. In Oklahoma all the gas-producing companies have voluntarily agreed to prorate their gas under the supervision of the corporation commission and have waived whatever legal rights they may have claimed.

FEDERAL REGULATIONS RELATING TO RESTRICTED INDIAN LANDS.

The following regulations,* aimed to prevent waste of oil and gas on restricted Indian lands in the State of Oklahoma, were promulgated by the Secretary of the Interior on October 20, 1915.

REGULATIONS TO GOVERN OIL AND GAS OPERATIONS ON RESTRICTED INDIAN LANDS IN OKLAHOMA.

DEFINITIONS.

The following expressions, wherever used in the lease and regulations, shall have the meaning now designated, viz:

Superintendent.—The superintendent of any Indian agency in Oklahoma, or any other person who may be in charge of such agency and reservation, and it shall be his duty to enforce compliance with these regulations.

Inspector.—Any person appointed as inspector of oil and gas operations, or who may be designated by the Secretary of the Interior or the Commissioner of Indian Affairs to supervise oil or gas operations on restricted Indian lands, acting under general instructions from the Bureau of Mines and under the supervision of the superintendent.

Oil lessee.—Any person, firm, or corporation to whom an oil-mining lease is made under these regulations.

Gas lessee.—Any person, firm, or corporation to whom a gas lease is made under these regulations.

Leased lands.—The term "leased lands" or "leased premises" or "leased tract" shall mean any restricted lands belonging to Indian allottees within the State of Oklahoma from which restrictions have not been removed, and which have been leased by such allottees with the approval of the Secretary of the Interior.

OPERATIONS.

1. No operations shall be permitted upon any tract of land until a lease covering such tract shall have been approved by the Secretary of the Interior.

POWERS AND DUTIES OF INSPECTOR.

It shall be the duty of the inspector—

2. To visit from time to time leased lands where oil and gas mining operations are being conducted and to inspect and supervise such operations, with a view

* Regulations to govern oil and gas operations on restricted Indian lands in Oklahoma: Dept. of the Interior, 1915, 8 pp.

to preventing waste of oil and gas, damage to oil, gas, or water bearing formations, or to coal measures or other mineral-bearing deposits, or injury to property or life, in accordance with the provisions of these regulations.

3. To make reports to the superintendent and to the Bureau of Mines as to the general conditions of the leases, property, and the manner in which operations are being conducted and his orders complied with.

4. To consult and advise with the superintendent as to the condition of the leased lands and to submit information and recommendations from time to time for safeguarding and protecting the property of the lessor and securing compliance with the provisions of these regulations.

5. To give such orders or notices as may be necessary to secure compliance with the regulations and to issue all necessary instructions or order to lessees to stop or modify such methods or practices as he may consider contrary to the provisions of such regulations.

6. To modify or prohibit the use or continuance of any operation or method which, in his opinion, is causing or is likely to cause any surface or underground waste of oil or gas or injury to any oil, gas, water, coal, or other mineral formation, or which is dangerous to life or property or in violation of the provisions of these regulations.

7. To prescribe, subject to the approval of the superintendent, the manner and form in which all records or reports called for by these regulations shall be made by the lessee.

8. To prohibit the drilling of any well into any producing sand, when in his opinion and with the approval of the superintendent the marketing facilities are inadequate, or insufficient provision has been made for controlling the flow of oil or gas reasonably to be expected therefrom, until such time as suitable provision can be made.

9. To prescribe or approve the methods of drilling wells through coal measures or other mineral deposits.

10. To determine when and under what conditions a producing well may be drilled deeper and under what conditions a producing well or sand may be abandoned.

11. To require that tests shall be made to detect waste of oil or gas or the presence of water in a well, and to prescribe or approve the methods of conducting such tests.

12. To require that any condition existing subsequent to the completion of a well which is causing, or is likely to cause, damage to an oil, gas, or water-bearing formation, or to coal measures, or other mineral deposits, or which is dangerous to life or property, be corrected as he may prescribe or approve.

13. To approve the type or size of separators used to separate the oil, gas, or water coming from a well.

14. The inspector may limit the percentage of the open flow capacity of any well which may be utilized when in his opinion such action is necessary to properly protect the gas-producing formation.

15. The inspector shall be the sole judge of whether his orders have been fully complied with and carried out.

DUTIES OF LESSEES.

16. Before actual drilling or development operations are commenced on the leased lands, or within not less than 30 days from the date of approval of these regulations in case of producing leases, or leased lands on which such operations have been commenced prior to such approval, the lessee or assignee

shall appoint a local or resident representative within Osage County or Oklahoma on whom the superintendent or other authorized representative of the Department of the Interior may serve notices or otherwise communicate with, in securing compliance with these regulations and shall notify the superintendent of the name and post-office address of the representative so appointed.

In the event of the incapacity or absence from the county of such designated local or resident representative, the lessee shall appoint some person to serve in his stead, and in the absence of such representative, or of notice of the appointment of a substitute, any employee of the lessee upon the leased premises or the contractor or other person in charge of drilling operations thereon shall be considered the representative of the lessee for the purpose of service of orders or notices as herein provided, and service upon any such employee, contractor, or other person shall be deemed service from the lessee.

17. Five days prior to the commencement of drilling operations lessee shall submit, on forms to be furnished by the superintendent, a report in duplicate showing the location of the proposed wells.

18. Lessee shall keep upon the leased premises accurate records of the drilling, redrilling, or deepening of all wells, showing formations drilled through, casing used, together with other information as indicated on prescribed forms to be furnished by the superintendent and shall transmit such and other reports of operations when required by the superintendent.

19. Lessee shall furnish on the 1st day of January and the 1st day of July of each year a plat in manner and form as prescribed by the superintendent, showing all wells, active or abandoned, on the leased lands, and other related information. Blank plats will be furnished upon application.

20. Lessee shall clearly and permanently mark all rigs or wells in a conspicuous place, with the name of the lessee and the number or designation of the well, and shall take all necessary precautions for the preservation of these markings.

21. Lessee shall not drill within 300 feet of boundary line of leased lands except with the consent of the superintendent. Lessee shall not locate any well or tank within 200 feet of any public highway or any building used as a dwelling, granary, barn, or established watering place, except with the written permission of the superintendent.

22. Lessee shall notify the superintendent, in advance, of intention to use the mud-fluid process of drilling, so that the inspector may approve the method and material to be used, in the event the operator is not familiar with this process.

23. Lessee shall provide a properly prepared slush pit into which all sand pumpings and other materials extracted from the well during the process of drilling shall be deposited. Such sand pumpings and materials shall not be allowed to run over the surface of the land. The construction of such pits shall be subject to the approval of the inspector.

24. Lessee shall effectually shut out and exclude all water from any oil or gas bearing stratum and take all proper precaution and measures to prevent the contamination or pollution of any fresh-water supply encountered in any well drilled for oil or gas.

25. Lessee shall protect to the satisfaction of the inspector each productive oil or gas bearing formation drilled through for the purpose of producing oil or gas from a lower formation.

26. Lessee shall place an approved gate valve, or other approved controlling device, on the innermost string of casing seated in the well, and keep same in place and in proper condition for use until the well is completed, whenever

drilling operations are commenced in "wildcat" territory, or in a gas or oil field where high pressures are known to exist, whenever the inspector shall deem same necessary for the proper control of the production from the well.

27. When natural gas is encountered in commercial quantities in any well, lessee shall confine such gas to its natural stratum until such time as the same can be produced and utilized without waste, it being understood that a commercial quantity of gas produced by a well is any unrestricted flow of natural gas in excess of 2,000,000 cubic feet per 24 hours: *Provided*, That if in the opinion of the superintendent gas of a lesser quantity shall be of commercial value the superintendent shall have authority to require the conservation of said gas. Water shall not be introduced into any well where such introduction will operate to kill or restrict the open flow of gas therein.

28. Lessee shall separate the oil from the gas when both are produced in commercial quantities from the same formation or under such conditions as might result in waste of oil or gas in commercial quantities.

29. Lessee shall not use natural gas from a distinct or separate stratum for the purpose of flowing or lifting the oil.

30. Lessee shall prevent oil or gas, or both, from escaping from any well into the open air, and not permit any oil or gas well to go wild or to burn wastefully.

31. Lessee shall not use natural gas in place of steam to operate engines or pumps under direct pressure except with the special permission of the inspector.

32. Lessee shall not use natural gas in flambeau lights, save as authorized or approved by the inspector.

33. Lessee shall use every possible precaution, in accordance with the most approved methods, to stop and prevent waste of natural gas and oil, or both, at the wells and from connecting lines, and to prevent the wasteful utilization of such gas about the well.

34. Lessee shall notify the superintendent a reasonable time in advance of starting work of intention to redrill, deepen, plug, or abandon a well; and whenever the superintendent or inspector has given notice that extra precautions are necessary in the plugging of wells in a particular territory, lessee shall give at least three days' advance notice of such intended plugging.

35. Lessee shall not abandon any well for the purpose of drilling deeper for oil or gas unless the producing stratum is properly protected, and shall not abandon any well producing oil or gas except with the approval of the superintendent or where it can be demonstrated that the further operation of such well is commercially unprofitable.

36. Lessee shall plug and fill all dry or abandoned wells on the leased lands in the manner required, and where any such well penetrates an oil or gas bearing formation it shall be thoroughly cleaned to the bottom of the hole before being plugged or filled, and shall then be filled with mud-laden fluid of a consistency approved by the inspector, from the bottom to the top thereof, before any casing is removed from the well, or in lieu of the use of such mud fluid, each oil and gas bearing formation shall be adequately protected by cement, or approved plugs, or by both such plugs and cement, and the well filled in above and below such cement or plugs with material approved by the inspector.

Where both fresh water and salt water are encountered in any dry or abandoned well which is not being filled with mud-laden fluid as hereinbefore provided, the fresh water shall be sufficiently protected against contamination by cement or approved plugs, or by both such cement and plugs, to be placed at such points in the well as the inspector shall approve for the protection of the fresh water.

37. If such abandoned or dry well be in a coal bed or other mineral vein deposit or be in such condition as to warrant taking extraordinary precautions, the inspector may require such variations in the above-prescribed methods of plugging and filling as may be necessary in his judgment to protect such seam or deposit against infiltration of gas or water and to protect all other strata encountered in the well.

38. The manner in which such mud-laden fluid, cement, or plugs shall be introduced into any well being plugged, and the type of plugs so used, shall be subject to the approval of the inspector.

In the event the lessee or operator shall fail to plug properly any dry or abandoned well in accordance with these regulations, the superintendent may, after five days' notice to the parties in interest, plug such well at the expense of the lessee or his surety.

39. All B-S or water from tanks or wells shall be drained off into proper receptacles located at a safe distance from tanks, wells, or buildings, to the end that same may be disposed of by being burned or transported from the premises.

Where it is impossible to burn the B-S, or where it is necessary to pump salt water in such quantities as would damage the surface of the leased land or adjoining property, or pollute any fresh water, the lessee shall notify the superintendent, who shall give instructions in each instance as to the disposition of such B-S or salt water.

40. Lessee shall make a full and complete report to the superintendent of all accidents or fires occurring on the leased premises.

41. Lessee shall provide approved tankage of suitable shape for accurate measurement, into which all production of crude oil shall be run from the wells, and shall furnish the superintendent copies of accurate tank tables and all run tickets, as and when requested.

42. The superintendent may make arrangements with the purchasers of oil for the payment of the royalty, but such arrangements, if made, shall not relieve the lessee from responsibility for the payment of the royalty should such purchaser fail, neglect, or refuse to pay the royalty when it becomes due: *Provided*, That no oil shall be run to any purchaser or delivered to the pipe line or other carrier for shipment or otherwise conveyed or removed from the leased premises until a division order is executed, filed, and approved by the superintendent, showing that the lessee has a regularly approved lease in effect and the conditions under which the oil may be run. Lessees shall be required to pay for all oil or gas used off the leased premises for operating purposes; affidavits shall be made as to the production used for such purposes and royalty paid in the usual manner. The lessee or his representatives shall be present when oil is taken from the leased premises under any division order, and will be responsible for the correct measurement thereof and shall report all oil so run.

The lessee shall also authorize the pipe-line company or the purchaser of oil to furnish the superintendent with a monthly statement, not later than the 10th day of the following calendar month, of the gross barrels run as common carrier shipment or purchased from his lease or leases.

43. Lessee will not be permitted to use any timber from any Osage lands except under written agreement with the owner, and in all cases where lands are restricted such agreement shall be subject to the approval of the superintendent or inspector. Lessee shall, when requested by the superintendent, furnish a statement under oath as to whether the rig timbers were purchased on the leased tract, and if so, state the name of the person from whom purchased

and give such other information regarding the procurement of timber as the superintendent may desire.

44. Unless expressly provided for in the lease, lessee shall pay to the superintendent for the parties in interest all reasonable damage done to the surface and any growing crops thereon or to improvements on said land in the amount of such damage when agreed upon between the parties in interest. When such amount can not be agreed upon, any of such parties may notify the superintendent, whereupon the superintendent shall notify the parties in interest that if such claims can not be arbitrated satisfactorily, he will, after 10 days from date of notice, investigate the matter of damage, such notice to be sent the lessee, allottee, or his heirs, and such other person as may have informed the superintendent in writing of a claim to an interest in such lands. The superintendent shall thereupon determine the damage and apportionment thereof between the parties in interest, such determination to be final unless an appeal therefrom be taken to the Secretary of the Interior within 10 days from the date of notice of such determination. The decision of the Secretary of the Interior shall be final and conclusive upon all parties concerned. The lessee shall be permitted to proceed with operations pending determination of the amount of damage by the superintendent upon depositing with the superintendent such amount as he may stipulate as sufficient to cover the damages claimed, and such lessee may continue with operations pending appeal upon depositing such additional amount, if any, as may be sufficient to cover the damages as fixed and apportioned by the superintendent, the surplus, if any, to be returned to the lessee. Pending action upon the appeal so much of said amount as is not in dispute by the parties in interest may be disbursed.

45. Failure to comply with any provision of these regulations shall subject the lease to cancellation by the Secretary of the Interior or the lessee to a fine of not more than \$500 per day for each and every day the terms of the lease or of the regulations are violated, or the orders of the superintendent pertaining thereto are not complied with, or to both such fine and cancellation, in the discretion of the Secretary of the Interior; *Provided*, That the lessee shall be entitled to notice and hearing with respect to the terms of the lease or of the regulations violated, which hearing shall be held by the superintendent, whose finding shall be conclusive unless an appeal be taken to the Secretary of the Interior within 30 days after notice of the superintendent's decision, and the decision of the Secretary of the Interior upon appeal shall be conclusive.

46. These regulations shall become effective and in full force from and after the date of approval and shall be subject to change or alteration at any time by the Secretary of the Interior.

E. B. MERITT,

Assistant Commissioner of Indian Affairs.

VAN. H. MANNING,

Director Bureau of Mines.

Approved October 20, 1915, DEPARTMENT OF THE INTERIOR.

FRANKLIN K. LANE.

STATE LAWS TO PREVENT WASTE OF NATURAL GAS AND OIL.

NATURAL GAS.

An act^a approved March 30, 1915, provides for preventing waste of natural gas in the State, for prorating the gas, confers authority on

^a House bill No. 395, Session Laws of Oklahoma, 1915, ch. 197, pp. 326-329.

the corporation commission to enforce the act, and prescribes a penalty for its violation. The act reads as follows:

An Act To conserve natural gas in the State of Oklahoma, to prevent waste thereof, providing for the equitable taking and purchase of same, conferring authority on the corporation commission, prescribing a penalty for violation of this act, repealing certain acts, and declaring an emergency.

Be it enacted by the people of the State of Oklahoma:

WASTE PROHIBITED.

SECTION 1. That the production of natural gas in the State of Oklahoma in such manner, and under such conditions, as to constitute waste, shall be unlawful.

WASTE DEFINED.

SEC. 2. That the term waste, as used herein, in addition to its ordinary meaning, shall include escape of natural gas in commercial quantities into the open air, the intentional drowning with water of a gas stratum capable of producing gas in commercial quantities, underground waste, the permitting of any natural-gas well to wastefully burn, and the wasteful utilization of such gas.

CONSERVATION OF GAS.

SEC. 3. That whenever natural gas in commercial quantities or a gas-bearing stratum, known to contain natural gas in such quantity, is encountered in any well drilled for oil or gas in this State, such gas shall be confined to its original stratum until such time as the same can be produced and utilized without waste, and all such strata shall be adequately protected from infiltrating waters. Any unrestricted flow of natural gas in excess of two million cubic feet per twenty-four hours shall be considered a commercial quantity thereof; *Provided*, That if in the opinion of the corporation commission gas of a lesser quantity shall be of commercial value, said commission shall have authority to require the conservation of said gas in accordance with the provisions of this act; *And, provided, further*, The gage of the capacity of any gas well shall not be taken until such well has been allowed an open flow for the period of three days.

EXCESS GAS SUPPLY—APPORTIONMENT.

SEC. 4. That whenever the full production from any common source of supply of natural gas in this State is in excess of the market demands, then any person, firm, or corporation, having the right to drill into and produce gas from any such common source of supply, may take therefrom only such proportion of the natural gas that may be marketed without waste, as the natural flow of the well or wells owned or controlled by any such person, firm, or corporation bears to the total natural flow of such common source of supply, having due regard to the acreage drained by each well, so as to prevent any such person, firm, or corporation securing any unfair proportion of the gas therefrom; *Provided*, That the corporation commission may by proper order permit the taking of a greater amount whenever it shall deem such taking reasonable or equitable. The said commission is authorized and directed to prescribe rules and regulations for the determination of the natural flow of any such well or wells and to regulate the taking of natural gas from any or all such common sources of supply within the State, so as to prevent waste, protect the interests of the public, and of all those having a right to produce therefrom, and to prevent unreasonable discrimination in favor of any one such common source of supply as against another.

"COMMON PURCHASER"—FAIR TREATMENT.

SEC. 5. That every person, firm, or corporation, now or hereafter engaged in the business of purchasing and selling natural gas in this State, shall be a common purchaser thereof, and shall purchase all the natural gas which may be offered for sale, and which may reasonably be reached by its trunk line, or gathering lines without discrimination in favor of one producer as against another, or in favor of any one source of supply as against another save as authorized by the corporation commission after due notice and hearing; but if any such person, firm, or corporation shall be unable to purchase all the gas so offered, then it shall purchase natural gas from each producer ratably. It shall be unlawful for any such common purchaser to discriminate between like grades and pressures of natural gas, or in favor of its own production, or of production in which it may be directly or indirectly interested, either in whole or in part, but for the purpose of prorating the natural gas to be marketed, such production shall be treated in like manner as that of any other producer or person, and shall be taken only in the ratable proportion that such production bears to the total production available for marketing. The corporation commission shall have authority to make regulations for the delivery, metering, and equitable purchasing and taking of all such gas and shall have authority to relieve any such common purchaser, after due notice and hearing, from the duty of purchasing gas of an inferior quality or grade.

HEARINGS BEFORE CORPORATION COMMISSION—HOW CONDUCTED.

SEC. 6. That any person, firm, or corporation, or the attorney general, on behalf of the State may institute proceedings before the corporation commission, or apply for a hearing before said commission, upon any question relating to the enforcement of this act; and jurisdiction is hereby conferred upon said commission to hear and determine the same, said commission shall set a time and place when such hearing shall be had and give reasonable notice thereof to all persons or classes interested therein by publication in some newspaper or newspapers having general circulation in the State, and shall in addition thereto cause notice to be served in writing upon any person, firm or corporation, complained against in the manner now provided by law for serving summons in legal actions. In the exercise and enforcement of such jurisdiction said commission is authorized to summon witnesses, make ancillary orders, and use such means and final process including inspection and punishment as for contempt, analogous to proceedings under its control over public service corporations as now provided by law.

APPEALS TO SUPREME COURT.

SEC. 7. That appellate jurisdiction is hereby conferred upon the supreme court of this State to review the orders of said commission made under this act. Such appeal may be taken by any person, firm, or corporation, shown by the record to be interested therein, in the same manner and time as appeals are allowed by law from other order of the corporation commission. Said orders so appealed from, may be superseded by the commission or by the supreme court upon such terms and conditions as may be just and equitable.

POWER OF CORPORATION COMMISSION—RULES AND REGULATIONS.

SEC. 8. That the corporation commission shall have authority to make regulations for the prevention of waste of natural gas, and for the protection of all

natural gas, fresh water, and oil-bearing strata encountered in any well drilled for oil or natural gas, and to make such other rules and regulations, and to employ or appoint such agents, with the consent of the governor, as may be necessary to enforce this act.

ACCEPTANCE BY PIPE LINES.

SEC. 9. Before any person, firm, or corporation shall have, possess, enjoy, or exercise the right of eminent domain, right of way, right to locate, maintain, construct, or operate pipe lines, fixtures, or equipments belonging thereto or used in connection therewith, for the carrying or transportation of natural gas, whether for hire or otherwise, or shall have the right to engage in the business of purchasing, piping, or transporting natural gas, as a public service, or otherwise, such person, firm, or corporation, shall file in the office of the corporation commission a proper and explicit authorized acceptance of the provisions of this act.

DUTIES OF MINE INSPECTOR UNCHANGED.

SEC. 10. That nothing contained in this act shall be construed to interfere with any duties now imposed by law upon the chief mine inspector of the State or his deputies.

VALIDITY OF SEVERAL SECTIONS OF ACT INDEPENDENT.

SEC. 11. That the invalidity of any section, subdivision, clause, or sentence of this act shall not in any manner affect the validity of the remaining portion thereof.

PENALTIES FOR VIOLATION.

SEC. 12. That in addition to any penalty that may be imposed by the corporation commission for contempt, any person, firm, or corporation, or any officer, agent, or employee thereof, directly or indirectly violating the provisions of this act, shall be guilty of a misdemeanor, and upon conviction thereof, in a court of contempt jurisdiction, shall be punished by a fine in any sum not to exceed \$5,000 or by imprisonment in the county jail not to exceed thirty days, or by both such fine and imprisonment.

EMERGENCY.

SEC. 13. For the preservation of the public peace, health, and safety, an emergency is hereby declared to exist, by reason whereof this act shall take effect and be in force from and after its passage and approval.

Approved March 30, 1915.

CRUDE OIL OR PETROLEUM.

An act^a passed by the State legislature on February 11, 1915, to prevent waste of oil, to provide for equitable production, and conferring authority on the corporation commission for its enforcement reads as follows:

^a House bill No. 168, Session Laws of Oklahoma, 1915, chap. 25, pp. 28-31.

An Act Defining and prohibiting the waste of crude oil or petroleum, providing for the equitable taking of the same from the ground and conferring authority on the corporation commission, prescribing the penalty for the violation of this act, and declaring an emergency.

Be it enacted by the people of the State of Oklahoma:

WASTE PROHIBITED.

SECTION 1. That the production of crude oil or petroleum in the State of Oklahoma, in such manner and under such conditions as to constitute waste is hereby prohibited.

PRODUCTION AND SALE REGULATED—CORPORATION COMMISSION.

SEC. 2. That the taking of crude oil or petroleum from any oil-bearing sand or sands in the State of Oklahoma at a time when there is not a market demand therefor at the well at a price equivalent to the actual value of such crude oil or petroleum is hereby prohibited, and the actual value of such crude oil or petroleum at any time shall be the average value as near as may be ascertained in the United States at retail of the by-products of such crude oil or petroleum when refined, less the cost and a reasonable profit in the business of transporting, refining, and marketing the same, and the corporation commission of this State is hereby invested with the authority and power to investigate and determine from time to time the actual value of such crude oil or petroleum by the standard herein provided; and when so determined, said commission shall promulgate its findings by its orders duly made and recorded and publish the same in some newspaper of general circulation in the State.

WASTE DEFINED—PROTECTION.

SEC. 3. That the term "waste" as used herein, in addition to its ordinary meaning, shall include economic waste, underground waste, surface waste, and waste incident to the production of crude oil or petroleum in excess of transportation or marketing facilities or reasonable market demands. The corporation commission shall have authority to make rules and regulations for the prevention of such wastes, and for the protection of all fresh-water strata, and oil and gas bearing strata, encountered in any well drilled for oil.

PRODUCTION REGULATED—DISCRIMINATION OF PURCHASER PROHIBITED.

SEC. 4. That whenever the full production from any common source of supply of crude oil or petroleum in this State can only be obtained under conditions constituting waste, as herein defined, then any person, firm, or corporation having the right to drill into and produce oil from any such common source of supply may take therefrom only such proportion of all crude oil and petroleum that may be produced therefrom, without waste, as the production of the well or wells of any such person, firm, or corporation bears to the total production of such common source of supply. The corporation commission is authorized to so regulate the taking of crude oil or petroleum from any or all such common sources of supply, within the State of Oklahoma, as to prevent the inequitable or unfair taking, from a common source of supply, of such crude oil or petroleum, by any person, firm, or corporation, and to prevent unreasonable discrimination in favor of any one such common source of supply as against another.

WELLS GAGED—GOVERNOR TO CONSENT.

SEC. 5. That for the purpose of determining such production, a gage of each well shall be taken under rules and regulations to be prescribed by the cor-

poration commission, and said commission is authorized and directed to make and promulgate, by proper order, such other rules and regulations and to employ or appoint such agents, with the consent of the governor, as may be necessary to enforce this act.

ENFORCEMENT OF ACT—HEARINGS BEFORE CORPORATION COMMISSION.

SEC. 6. That any person, firm, or corporation, or the attorney general on behalf of the State, may institute proceedings before the corporation commission, or apply for a hearing before said commission, upon any question relating to the enforcement of this act, and jurisdiction is hereby conferred upon said commission to hear and determine the same. Said commission shall set a time and place, when and where such hearing shall be had and give reasonable notice thereof to all persons or classes interested therein, by publication in some newspaper or newspapers, having general circulation in the State, and in addition thereto, shall cause reasonable notice in writing to be served personally on any person, firm, or corporation complained against. In the exercise and enforcement of such jurisdiction, said commission is authorized to determine any question or fact, arising hereunder, and to summon witnesses, make ancillary orders, and use mesne and final process, including inspection and punishment as for contempt, analogous to proceedings under its control over public-service corporations, as now provided by law.

APPEALS TO SUPREME COURT—EFFECT ON ORDERS.

SEC. 7. That appellate jurisdiction is hereby conferred upon the supreme court in this State to review the action of said commission in making any order, or orders, under this act. Such appeal may be taken by any person, firm, or corporation, shown by the record to be interested therein, in the same manner and time as appeals are allowed by law from other orders of the corporation commission. Said orders so appealed from shall not be superseded by the mere fact of such appeal being taken, but shall be and remain in full force and effect until legally suspended or set aside by the supreme court.

PENALTY FOR VIOLATION.

SEC. 8. That in addition to any penalty that may be imposed by the corporation commission for contempt, any person, firm, or corporation, or any officer, agent, or employee thereof, directly or indirectly violating the provisions of this act shall be guilty of a misdemeanor, and upon conviction thereof, in a court of competent jurisdiction, shall be punished by a fine in any sum not to exceed \$5,000, or by imprisonment in the county jail not to exceed thirty days, or by both fine and imprisonment.

STATE MAY SECURE RECEIVER—EXTENT AND MANNER.

SEC. 9. That in addition to any penalty imposed under the preceding section, any person, firm, or corporation, violating the provisions of this act, shall be subject to have his or its producing property placed in the hands of a receiver by a court of competent jurisdiction, at the suit of the State through the attorney general or any county attorney, but such receivership shall only extend to the operating of producing wells and the marketing of the production thereof, under the provisions of this act.

VALIDITY OF RELATIVE SECTIONS OF ACT.

SEC. 10. That the validity of any section, subdivision, clause, or sentence of this act shall not in any manner affect the validity of the remaining portion thereof.

EMERGENCY.

SEC. 11. For the preservation of the public peace, health, and safety, an emergency is hereby declared to exist by reason whereof this act shall take effect and be in force from and after its passage and approval.

Approved February 11, 1915.

52432°—Bull. 134—16—6

PUBLICATIONS ON PETROLEUM TECHNOLOGY.

A limited supply of the following publications of the Bureau of Mines is temporarily available for free distribution. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

BULLETIN 88. The condensation of gasoline from natural gas, by G. A. Burrell, F. M. Seibert, and G. G. Oberfell. 1915. 106 pp., 6 pls., 18 figs.

TECHNICAL PAPER 10. Liquefied products from natural gas, their properties and uses, by I. C. Allen and G. A. Burrell. 1912. 23 pp.

TECHNICAL PAPER 32. The cementing process of excluding water from oil wells, as practiced in California, by Ralph Arnold and V. R. Garfias. 1913. 12 pp., 1 fig.

TECHNICAL PAPER 37. Heavy oil as fuel for internal-combustion engines, by I. C. Allen. 1913. 36 pp.

TECHNICAL PAPER 38. Wastes in the production and utilization of natural gas and means for their prevention, by Ralph Arnold and F. G. Clapp. 1913. 29 pp.

TECHNICAL PAPER 42. The prevention of waste of oil and gas from flowing wells in California, with a discussion of special methods used by J. A. Pollard, by Ralph Arnold and V. R. Garfias. 1913. 15 pp., 2 pls., 4 figs.

TECHNICAL PAPER 45. Waste of oil and gas in the Mid-Continent fields, by R. S. Blatchley. 1914. 54 pp., 2 pls., 15 figs.

TECHNICAL PAPER 49. The flash point of oils; methods and apparatus for its determination, by I. C. Allen and A. S. Crossfield. 1913. 31 pp., 2 figs.

TECHNICAL PAPER 57. A preliminary report on the utilization of oil and natural gas in Wyoming, by W. R. Calvert, with a discussion of the suitability of natural gas for making gasoline, by G. A. Burrell. 1913. 23 pp.

TECHNICAL PAPER 66. Mud-laden fluid applied to well drilling, by J. A. Pollard and A. G. Heggem. 1914. 21 pp., 12 figs.

TECHNICAL PAPER 68. Drilling wells in Oklahoma by the mud-laden fluid method, by A. G. Heggem and J. A. Pollard. 1914. 27 pp., 5 figs.

TECHNICAL PAPER 70. Methods of oil recovery as practiced in California, by Ralph Arnold and V. R. Garfias. 1914. 57 pp., 7 figs.

TECHNICAL PAPER 72. Problems of the petroleum industry; results of conferences at Pittsburgh, Pa., August 1 and September 10, 1913, by I. C. Allen. 1914. 29 pp.

TECHNICAL PAPER 74. Physical and chemical properties of the petroleum of California, by I. C. Allen, W. A. Jacobs, A. S. Crossfield, and R. R. Matthews. 1914. 38 pp., 1 fig.

TECHNICAL PAPER 79. Electric lights for oil and gas wells, by H. H. Clark. 1914. 8 pp.

TECHNICAL PAPER 104. Analysis of natural gas and illuminating gas by fractional distillation in a vacuum at low temperatures and pressures, by G. A. Burrell, F. M. Seibert, and I. W. Robertson. 1915. 41 pp., 7 figs.

TECHNICAL PAPER 117. Quantity of gasoline necessary to produce explosive vapors in sewers, by G. A. Burrell and H. T. Boyd. 1916. 18 pp., 4 figs.

TECHNICAL PAPER 127. Hazards in handling gasoline, by G. A. Burrell. 1915. 12 pp.

TECHNICAL PAPER 130. Underground wastes in oil and gas fields and methods of prevention, by W. F. McMurray and J. O. Lewis. 1916. 31 pp., 1 pl., 9 figs.

TECHNICAL PAPER 131. The compressibility of natural gas at high pressures, by G. A. Burrell and I. W. Robertson. 1916. 12 pp., 2 figs.

TECHNICAL PAPER 161. Construction and operation of a single-tube cracking furnace for making gasoline, by C. P. Bowle. 1916. 16 pp., 10 pls.

PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.

The editions for free distribution of the following Bureau of Mines publications are exhausted, but copies may be obtained by purchase from the Superintendent of Documents, Government Printing Office, Washington, D. C., or can be consulted at public libraries. Prepayment of the price is required and should be made in cash (exact amount) or by postal or express money order payable to the superintendent of documents.

The Superintendent of Documents is an official of the Government Printing Office and is not connected with the Bureau of Mines.

BULLETIN 65. Oil and gas wells through workable coal beds; papers and discussions, by G. S. Rice, O. P. Hood, and others. 1913. 101 pp., 1 pl., 11 figs. 10 cents.

BULLETIN 114. Manufacture of gasoline and benzene-toluene from petroleum and other hydrocarbons, by W. F. Rittman, C. B. Dutton, and E. W. Dean, with a bibliography compiled by M. S. Howard. 1916. 268 pp., 9 pls., 45 figs. 35 cents.

TECHNICAL PAPER 3. Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp. 5 cents.

TECHNICAL PAPER 25. Methods for the determination of water in petroleum and its products, by I. C. Allen and W. A. Jacobs. 1912. 13 pp., 2 figs. 5 cents.

TECHNICAL PAPER 26. Methods for the determination of the sulphur content of fuels, especially petroleum products, by I. C. Allen and I. W. Robertson. 1912. 13 pp., 1 fig. 5 cents.

TECHNICAL PAPER 36. The preparation of specifications for petroleum products, by I. C. Allen. 1913. 12 pp. 5 cents.

TECHNICAL PAPER 51. Possible causes of the decline of oil wells, and suggested methods of prolonging yield, by L. G. Huntley. 1913. 32 pp., 9 figs. 5 cents.

TECHNICAL PAPER 53. Proposed regulations for the drilling of oil and gas wells, with comments thereon, by O. P. Hood and A. G. Heggem. 1913. 28 pp., 2 figs. 5 cents.

TECHNICAL PAPER 109. Composition of the natural gas used in 25 cities, with a discussion of the properties of natural gas, by G. A. Burrell and G. G. Oberfell. 1915. 22 pp. 5 cents.

TECHNICAL PAPER 115. Inflammability of mixtures of gasoline vapor and air, by G. A. Burrell and H. T. Boyd. 1915. 18 pp., 2 figs. 5 cents.

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Bulletin 135

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

COMBUSTION OF COAL AND DESIGN OF FURNACES

BY

HENRY KREISINGER, C. E. AUGUSTINE

AND

F. K. OVITZ



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COMBUSTION OF COAL AND DESIGN OF FURNACES.

By HENRY KREISINGER, C. E. AUGUSTINE, and F. K. OVITZ.

INTRODUCTION.

The Bureau of Mines is conducting investigations to determine how fuels belonging to or for the use of the United States Government can be utilized with greater efficiency. As a result of these investigations a number of reports have been published, each report treating of one phase of fuel utilization. A list of these publications is given at the end of this bulletin.

The efficient utilization of coal depends on the correct design of furnaces and on their proper operation. To attain both of these conditions a knowledge of the fundamental principles of combustion is necessary. Therefore, the study of the combustion of coal in industrial furnaces has been made a part of the bureau's fuel investigations.

The desirability of investigating the fundamental principles of combustion was suggested by the study of a large number of steaming tests made by the fuel-testing division on coals from all of the more important fields in the United States. The results of these tests have been published in Bulletin 325^a of the United States Geological Survey and in Bulletin 23^b of the Bureau of Mines.

As a result of that study a plan to investigate the combustion of coal in a special apparatus in which all conditions could be controlled was presented to the National Advisory Board for Fuels and Structural Materials at its meeting of April 10 and 11, 1906. The plan received the approval of the board, and since the establishment of the experiment station of the Bureau of Mines at Pittsburgh the investigation of the combustion of coal has been carried on intermittently. The history of these investigations is given in Technical Paper 63.^c

^a Breckenridge, L. P., A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., in 1904, 1905, and 1906: U. S. Geol. Survey Bull. 325, 1907, 196 pp.

^b Breckenridge, L. P., Kreisinger, Henry, and Ray, W. T., Steaming tests of coal and related investigations, Sept. 1, 1904, to Dec. 31, 1908: Bull. 23, Bureau of Mines, 1912, 380 pp.

^c Clement, J. K., Frazer, J. C. W., and Augustine, C. E., Factors governing the combustion of coal in boiler furnaces: Tech. Paper 63, Bureau of Mines, 1914, 46 pp.

PERSONNEL.

The combustion investigations reported in this bulletin were conducted by one fireman and five observers, three of whom were chemists collecting and analyzing samples of gases. The following men were at various times members of the crew: W. T. Ray, H. Kreisinger, J. K. Clement, J. C. W. Frazer, E. J. Hoffman, C. E. Augustine, F. K. Ovitz, A. E. Hall, L. L. Satler, A. L. Smith, V. B. Bonny, E. T. Gregg, L. H. Adams, W. L. Egy, and J. P. Stein.

Chemical analyses of coal and refuse were made under the direction of A. C. Fieldner, chemist, of the Bureau of Mines.

PURPOSE AND SCOPE OF COMBUSTION INVESTIGATIONS.

In practically all industrial furnaces the combustion of coal takes place in two stages—(1) combustion in the fuel bed, which includes the distillation of volatile matter and partial combustion or gasification of the fixed carbon; and (2) combustion of the gaseous and other combustible rising from the fuel bed in the combustion space.

The processes of combustion in a hand-fired furnace can be best explained by reference to figure 1. In the figure the curves show the percentages of the different gases at various points in the fuel bed and in the combustion space. The changes in the percentage of each gas indicate the process of combustion. The fuel bed is shown to be 6 inches thick. The oxygen (O_2) is all used at about $3\frac{1}{2}$ inches from the grate. At the same point the carbon dioxide (CO_2) reaches a maximum of about 12 per cent. Beyond this point the percentage of CO_2 drops and the percentage of carbon monoxide (CO) and other combustible increases rapidly, showing that the CO_2 is reduced by contact with hot carbon to CO. At the surface of the fuel bed the gases contain about 26 per cent of combustible, no oxygen, and about 8 per cent of CO_2 . Air is added over the fuel bed and the combustible is burned in the combustion space. At the end of 7 feet of the combustion space, the combustible gases are burned to 4 per cent and at the same time the CO_2 increases to 14 per cent. With a combustion space long enough the percentage of combustible would be reduced practically to zero.

The three main processes in the fuel bed are the oxidization of carbon to CO_2 , the reduction of CO_2 to CO, and the distillation of volatile matter. The zones where these three processes take place are indicated at the top of figure 1, and are called the oxidizing zone, the reducing zone, and the distillation zone. The boundaries between the three zones are not distinct, the zones merging gradually into one another.

The combustion investigations of the bureau are carried on in two parts. One part is the study of the processes of combustion in the fuel bed as affected by the rate of supplying air through the fuel bed, by

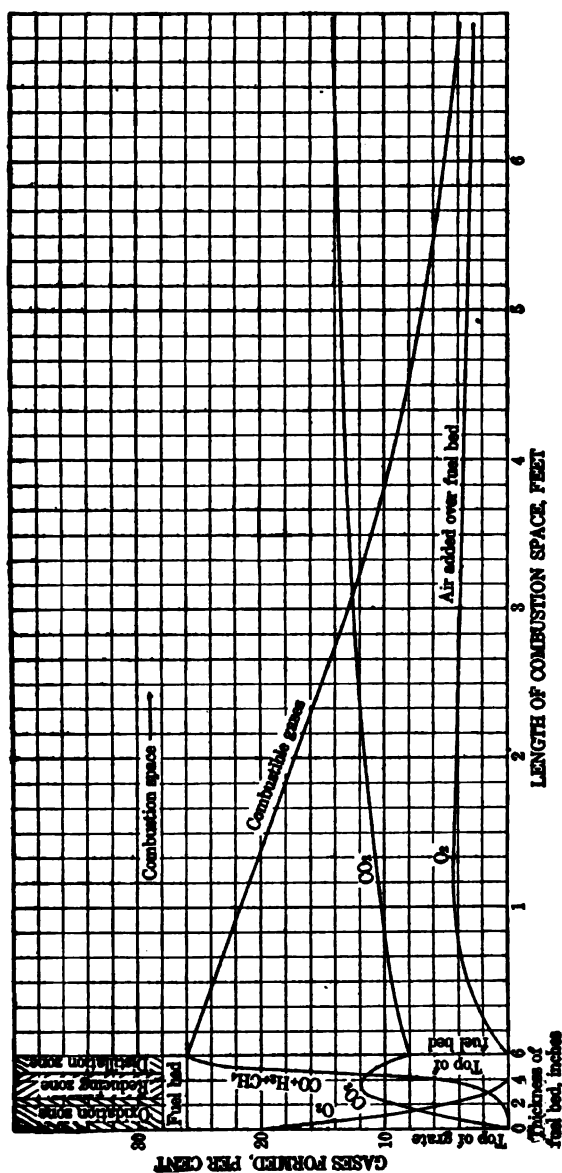


FIGURE 1.—Process of combustion of coal in a hand-fired furnace.

the character of the fuel as regards structure and composition, by the thickness of fuel bed, by the method of feeding the coal and the air, and by the method of heating the coal. Investigations so far completed show that the fuel bed in most industrial furnaces acts pri-

marily as a gas producer. The gases rising from a level fuel bed contain 15 to 32 per cent of combustible gases, about 8 per cent of carbon dioxide, and practically no free oxygen. This is true of even 6-inch fuel beds and rates of combustion as high as 120 pounds of coal per hour per square foot of grate. The process of combustion in the fuel bed is treated more thoroughly in Technical Paper 137.^a

The second part of the investigation is the study of the process of combustion of the gases and other combustible rising from the fuel bed in the combustion space, after a sufficient quantity of air has been added. The results of the study are given in this report.

The process of combustion in the combustion space is influenced by many factors, the most important of which are the following: The volume and shape of the combustion space; the kind of coal used, especially the character, and the amount of the volatile matter it contains; the rate of firing; the quantity of air supplied over the fuel bed; the rate of mixing the air with the combustible rising from the fuel bed; the rate of heating the coal; and the temperature in the combustion space.

The qualitative effects on the rate and completeness of combustion of many of these factors have been known for a long time. Thus, for example, it has been generally known that the higher the percentage of volatile matter in the coal and the higher the rate of combustion, the larger must be the combustion space to attain practically complete combustion. But the exact relation of these factors was not known; that is, the quantitative data presented in some convenient units, such as pounds, feet, or percentages, were lacking. It was to obtain such definite relations that the study of combustion in the space beyond the fuel bed was undertaken. On pages 68 to 89 are given data for proportioning the combustion spaces of industrial furnaces; these data, being the result of direct experimental work, furnish a more rational basis for design than has heretofore been available.

The apparatus so far used in the investigation was a special furnace having a combustion space 3 feet by 3 feet in cross section and about 40 feet long.

Provision was made to collect gas samples at about 5-foot intervals along the path of travel of the gases through the combustion space. Analyses of these samples furnished the principal data for the study of the process of combustion in the combustion space.

REASONS FOR SELECTING A SPECIAL FURNACE.

So many factors influence the process of combustion that it is impossible to get concordant results if all the factors vary at once. The apparatus and the experiments must be so planned that only

^a Kreslinger, Henry, Ovitz, F. K., and Augustine, C. E., *Combustion in the fuel bed of hand-fired furnaces*: Tech. Paper 137, Bureau of Mines, 1916, 76 pp.

one factor is varied at a time, while the other factors are fixed or made constant. The design of the apparatus must be such that a complete set of accurate observations can be taken. It was mainly for these reasons that the special furnace was chosen.

The first steps in the investigation were to study the effects of the size of the combustion space, of the rate of combustion, of the kind of coal, and of the air supply. The rate of heating, and the rate of mixing air added over the fuel bed with combustible gases were to be kept as constant as practicable. Only the normal mixing in the stoker and the natural diffusion of gases entered into the series of tests so far conducted. One reason for starting the investigation with this limited mixing was that most industrial furnaces operate under similar conditions. Another reason was to obtain time and experience for devising effective means for mixing and some convenient way of expressing the speed and thoroughness of mixing. It was realized at the time of planning these investigations that the rate of mixing of air added over the fuel bed with the combustible rising from the fuel bed greatly affects the rapidity of the combustion of the gases. The importance of mixing has been pointed out by Breckinridge.^a

Since the investigations recorded in this report were made the experimental furnace has been so modified that any quantity of air can be introduced over the fuel bed with controllable velocities and in four different directions, producing different rates of mixing. A series of tests has been started with this modified furnace with the object of determining the effect of mixing on the rapidity of combustion of the volatile matter of coal in the combustion space.

SCOPE OF THIS REPORT.

This report treats of the processes of combustion of the combustible substances rising from the fuel bed, which must be burned in the combustion space of the furnace. In other words, it considers the processes taking place in the furnace between the top of the fuel bed and the place where the gases leave the furnace. The knowledge of these processes makes it possible to determine the proper design of the combustion space of a furnace for burning a given coal under a given set of conditions.

This bulletin is a companion report to Technical Paper 137,^b and is a more complete report of the investigations, a part of which were reported in Technical Paper 63.^c

^a Breckinridge, L. P., A study of four hundred steaming tests made at the fuel-testing plant, St. Louis, Mo., in 1904, 1906, and 1908: U. S. Geol. Survey Bull. 325, 1907, p. 178.

^b Kreisinger, Henry, Ovitz, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces: Tech. Paper 137, Bureau of Mines, 1916, 76 pp.

^c Clement, J. K., Fraser, J. C. W., and Augustine, C. E., Factors governing the combustion of coal in boiler furnaces: Tech. Paper 63, Bureau of Mines, 1914, 46 pp.

The material presented consists mainly of the description of about 100 tests made in the special furnace having a long combustion space; the tabulated and plotted results of these tests; and the discussion of these results with deductions furnishing the basis for rational furnace design. The tests were made with three kinds of coal, namely, Pocahontas, Pittsburgh, and Illinois coal. The coals were burned at five rates—20, 30, 40, 50, and 60 pounds per square foot of grate per hour. Several tests were made with each coal at each rate of combustion, each test being run with a different percentage of excess air.

The material is arranged in four parts. The first part contains the description of the apparatus and the methods of conducting the experiments; the second part gives the results of the experiments, explains their meaning and points out their practical application; the third part consist of discussions of miscellaneous observations; and the fourth part contains a discussion of the process of combustion in the combustion space and of the laws that govern it.

PRACTICAL DEDUCTIONS FROM INVESTIGATIONS.

The practical deductions are given in figures 29, 30, 32, and 33 and the text on pages 68 to 89. The data shown in the figures may be used for proportioning the combustion space in designing furnaces. Although these data were obtained from experiments with a Murphy furnace and are therefore particularly applicable to furnaces of that type, it is believed that they may be a valuable guide in the proportioning of other furnaces. When applying the data to other furnaces the designer should give full consideration to the method of introducing secondary air and the facilities for mixing it with the combustible rising from the fuel bed. If the methods of introducing the air and the facilities for mixing it are better than those used in the experimental furnace, smaller combustion space than is indicated in the figures may be used. If, however, the methods are not as good the size of the combustion space should be increased. For the best results the secondary air should be introduced as near to the fuel bed as practicable, and the air should be supplied in a large number of streams at high velocities.

GENERAL SUMMARY.

The gases rising from the fuel bed of a Murphy stoker contain 10 to 28 per cent by volume of combustible. As the gases flow through the combustion space they mix with the air added over the fuel bed and burn. Because of this combustion the percentage of combustible decreases along the path of gases, the rate of decrease being rapid at first, but slowing down as the gases move farther from the fuel bed.

This combustion process is discussed on pages 54 to 61 in connection with figures 14 to 18, inclusive.

Inasmuch as the gases rising from the fuel bed contain 10 to 28 per cent of combustible and practically no free oxygen, additional air must be supplied over the fuel bed to insure complete combustion. In order that this additional air may flow into the furnace, the pressure of gases in the furnace must be somewhat below that of the outside air; in other words, there must be a "draft" over the fire. If the pressure in the furnace is nearly the same as that of the outside atmosphere, the additional air must be supplied under pressure with a blower or some other device. Therefore, a furnace can not be operated successfully with the so-called balanced draft, without supplying under pressure the necessary air over the fuel bed.

RELATION OF COMPLETENESS OF COMBUSTION TO SIZE OF COMBUSTION SPACE.

The combustible gases leaving the fuel bed represent 40 to 60 per cent of the total heat units in the coal, so that it can be said that, on an average, one-half of the combustion of coal takes place in the fuel bed and one-half in the combustion space. The completeness of combustion of the gases depends on the size of the combustion space. This feature of the combustion process is discussed in detail on pages 61 to 67 in connection with figures 19 to 23, inclusive.

RATE OF FEEDING AIR AND COAL.

The completeness of combustion with any given size of combustion space depends on the excess of air and the rate of firing the coal. In general, the smaller the excess of air and the higher the rate of firing the less completely are the furnace gases burned. The effect of these two factors on the completeness of combustion is discussed on page 68 in connection with figures 24 to 28, inclusive.

The application to furnace design of the relation between the completeness of combustion on the one hand and the size of combustion space, the excess of air and the rate of firing on the other, is given in pages 68 to 78 in connection with figures 29 and 30.

CHARACTER OF FUEL.

The effect of the composition of the coal on the required combustion space, when the coal is burned under a given set of conditions is discussed on pages 78 to 90 in connection with figures 31 and 32. In general, it can be said that for any given set of conditions as to rate of combustion and excess of air the required size of combustion space is approximately in direct proportion to the product of the percentage of the volatile matter times a factor representing its

quality. The indicator of the quality of the volatile matter is the ratio of the volatile carbon to available hydrogen. The more carbon in the volatile matter the more difficult the latter is to burn.

The size of the combustion space required for any set of conditions of combustion also appears to be directly proportional to the percentage of oxygen in the moisture-free and ash-free coal. However, it is very likely that the burning properties of the coal are not directly affected by the oxygen in the coal, but that the oxygen indicates the stage in carbonization of the original plant substance, and that this stage of carbonization, or the age of the coal, has a direct effect on burning properties.

The percentage of excess air that gives the best results in any steam-generating apparatus varies with the size of the furnace and the kind of fuel. In two furnaces burning the same fuel but having different sizes of combustion space, the one with the smaller combustion space will require more excess air for the best results than will the one with the larger combustion space. Also of two furnaces exactly alike in size, but burning different coals, the one burning the coal lower in volatile matter and oxygen gives the best results with lower excess of air than is necessary for the best results in the furnace burning the coal higher in volatile matter and oxygen. This fact explains why in one plant the highest efficiency may be obtained with 14 per cent of CO_2 in the gases and in another plant with only 10 per cent of CO_2 . This feature is discussed in detail on pages 90 to 97 in connection with figures 35 to 39, inclusive.

STRATIFICATION OF GASES.

The gases in the furnace are not of uniform composition; they tend to flow through the combustion space in stratified streams. This tendency to stratification retards combustion and makes gas sampling difficult, and persists even with the apparently violent mixing of the Murphy furnace. It is impossible to draw definite conclusions of the progress of combustion by taking only one gas sample at any one cross section of the combustion space. Even by taking several simultaneous samples at different points in any one cross section it is difficult to arrive at the average composition of the gases. This statement is true in varying degrees of all commercial furnaces. This feature is discussed on pages 97 to 104, inclusive.

CONTENT OF CO_2 AND CO IN GASES.

There is a definite relation for each coal between the excess of air supply and the percentage of CO_2 in the furnace-gases. This relation is shown for Pocahontas, Pittsburgh, and Illinois coal in figure 40, and is discussed on page 105.

The total combustible gases at any cross section of the combustion space are proportional to the percentage of CO in the furnace gases. Approximately the percentage of CO forms four-fifths of the total combustible gases. The other one-fifth consists mainly of hydrogen; only a trace of methane is usually found in the gases. Therefore, by determining the percentage of CO in the furnace gases, the heat losses due to the escape of all combustible gases can be closely computed. The relation between the percentage of CO and these heat losses is shown in figures 41 and 42, and is discussed on pages 105 to 109.

EFFECT OF GAS SAMPLERS ON TEMPERATURE DROP.

The drop in temperature along the path of the furnace gases in the experimental furnace was mainly due to the heat absorption by the water-cooled gas samplers. The gas samplers were absorbing heat at the rate of one boiler horsepower on less than 0.25 square foot of exposed surface. This drop in temperature is discussed on pages 109 to 111.

BURNING OF VOLATILE MATTER.

The volatile matter leaves the fuel bed as complex hydrocarbon compounds. In the absence of sufficient oxygen for their complete combustion these hydrocarbons are quickly decomposed by the high furnace temperature into soot, hydrogen, and carbon monoxide. The formation of CO is due to the presence of CO₂ and the small supply of oxygen. At a distance of 1 or 2 feet from the surface of the fuel bed only a very small amount of hydrocarbons can be found in any state, gaseous, liquid, or solid. The solid substance present in the flames is mostly soot, with a trace of tars. This fact is shown in Table 11 and figure 44. All hydrocarbons are unstable at furnace temperature, and unless enough air to insure complete combustion is quickly mixed with them at the time they are distilled, they are quickly decomposed, the ultimate product consisting mostly of soot, H₂, and CO. Methane (CH₄), which is perhaps the most stable hydrocarbon, is found only in traces, 1 foot beyond the surface of the fuel bed. The persistence of CO in the furnace gases is not due to the difficulty of burning it, but to its constant formation by the reaction between soot and H₂ with CO₂. This is the reason why CO is found in the furnace gases after all other forms of combustible have practically disappeared. Under ordinary furnace conditions there is no need of analyzing for any other combustible gas when the CO content is less than 1 per cent.

Soot is formed at the surface of the fuel bed by heating the hydrocarbons in absence of air; it is not formed by the hydrocarbon gases striking the cooling surfaces of the boiler. As a matter of fact only a very small trace of the hydrocarbon gases ever reaches the surfaces of

the boiler; hydrocarbons that do so are prevented from decomposition by the cooling effect of the contact. The cooling surfaces do not cause the formation of soot, they merely collect soot and prevent its combustion. Hydrocarbons are stable at low temperatures but decompose readily at high temperatures.

Only when the furnace temperature is very low, say below $1,000^{\circ}\text{F.}$, is there found an appreciable quantity of hydrocarbons 2 feet from the fuel bed. This subject is discussed in detail on pages 112 to 125.

The rapidity of burning combustible in the combustion space depends on the quantity of oxygen intimately mixed with the combustible. The larger this quantity the faster the combustible burns. This phase of the combustion process is considered on pages 125 to 129.

Thorough mixing of the gases produces a uniform mixture of the oxygen and the combustible, thereby making possible the most rapid combustion with any given air supply. In a large number of instances the rapidity of combustion is limited by the rate of mixing. Thus mixing is a very important factor in the attainment of good combustion. Mixing is discussed in more detail on pages 129 to 132.

COMBUSTION TESTS.

GENERAL STATEMENT.

The investigation of the combustion in the space of a furnace above the fuel bed was carried on in the special furnace having a horizontal combustion chamber about 40 feet long and of nearly constant cross section. The furnace was fed with a Murphy stoker, and the products of combustion were discharged from the end of the combustion chamber under a Heine boiler. Air was supplied under the grate and over the fuel bed with a fan blower. An exhaust fan provided sufficient reduction of pressure at the discharge end of the furnace.

The data obtained in these experiments are based mainly on series of experiments in which several sets of gas samples were simultaneously collected from the combustion space at intervals of about 5 feet along the path of the gases. Furnace temperatures were also measured at the places where the gas samples were collected. Gas pressures under the grate, over the fuel bed, and at the discharge end of the furnace were determined periodically. The coal was weighed as it was supplied to the furnace.

DESCRIPTION OF APPARATUS.

The general external appearance of the experimental furnace and its connection to the Heine boiler are shown in Plate I. A horizontal and a vertical longitudinal section through the furnace are shown in figure 2. The furnace is essentially a brick tunnel 3 by 3 feet



EXPERIMENTAL FURNACE. CONNECTIONS TO HEINE BOILER SHOWN.

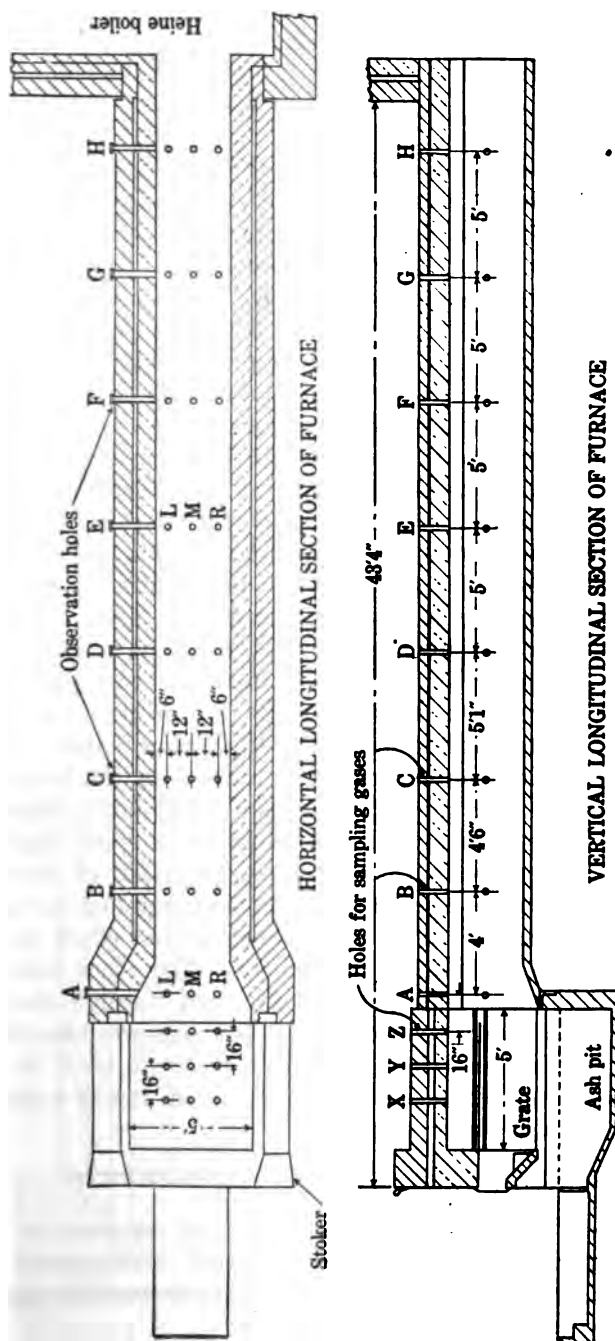


FIGURE 2.—Horizontal and vertical longitudinal sections through experimental furnaces. A to H are sections where gas samples were taken. Circles show position of sampling holes. L, M, and R indicate left, middle, and right sampler, respectively.

in cross section and about 40 feet long, the tunnel forming the combustion chamber. At one end of the chamber there is a standard Murphy stoker having 25 square feet of projected grate area. The other end of the tunnel connects with the combustion chamber of a hand-fired Heine boiler, the connection being made through the side wall of the boiler furnace.

The details of the walls and the roof are also shown in figure 2. The side walls consist of two walls separated by a 2-inch air space. The inner wall is built of fire brick and is 9 inches thick. The outer wall is of common red brick and is 8 inches thick. The roof consists of two brick arches separated by a 1-inch layer of asbestos cement; the inner arch is of fire brick and is 8 inches thick; the outer arch, which is only 4 inches thick, is built of pressed red brick. Figure 3 shows two sections through the stoker and illustrates the construction of the arch over the stoker. The arch contains an air space through which air is delivered to the tuyères supplying air over the fuel bed.

In one of the side walls are a number of observation holes for measuring temperatures, spaced about 5 feet apart along the length of the furnace. In the roof three holes for sampling gases were placed opposite each hole in the side wall for temperature measurements. The position of the holes is shown in Plate I and figure 2.

The difference of pressures necessary to move the gases through the furnace was obtained by an exhaust fan and two pressure fans. The exhaust fan, which was driven by a direct-connected steam engine, was situated at the uptake of the Heine boiler. The main object of connecting the experimental furnace to the boiler was to cool the products of combustion so they could be handled by the exhaust fan. The exhaust fan could be run at a speed high enough to maintain a pressure in the uptake of $1\frac{1}{2}$ inches of water below atmospheric pressure. The pressure fan that supplied air under the grate was direct-connected to a steam engine and could maintain a pressure of 1 inch of water in the ash pit. The other pressure fan, which helped to supply air over the fuel bed through the tuyères, was a small fan directly connected to a $\frac{1}{2}$ -horsepower electric motor.

The coal was delivered from a bin to a platform built in front of and over the stoker, where, as previously stated, it was weighed as it was supplied to the stoker magazine.

METHOD OF CONDUCTING EXPERIMENTS.

During about one-half of the investigations reported in this bulletin, the experimental furnace was operated continuously for producing steam for the power plant. When not so used fire was started in it one day before running a test.

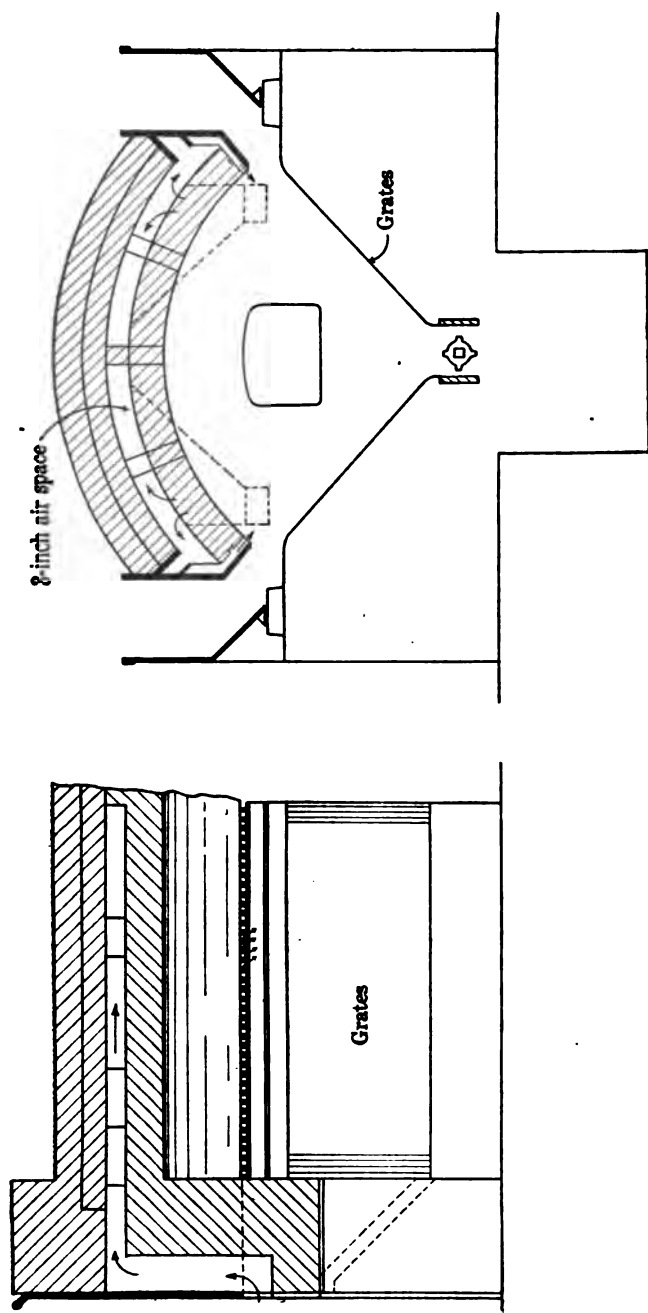


FIGURE 3.—Sections through the stoker, showing construction of the arch and method of supplying air to the tuyeres.

METHOD OF STARTING AND STOPPING THE TESTS.

Each test was started at about 10 to 11 o'clock a. m. and continued until 3 to 4.30 o'clock p. m. Before the test was started the fire was cleaned and all refuse removed from the ash pit. The test was closed with the same amount of coal in the magazines as there was at the time of starting.

CONDITIONS DURING THE TESTS.

During the test the rate of combustion, the thickness of fuel bed, and the air supply were kept as constant as practicable. The rate of combustion was controlled by weighing the coal as it was supplied to the magazines and by regulating the speed of the engine driving the feeding mechanism of the stoker and the speed of the pressure fan supplying air under the grate. The air supply was controlled by observing closely an automatic CO₂ recorder. Either the Bimeter CO₂ recorder or the Uehling CO₂ recorder^a was used for this purpose. The first few tests were controlled by analyzing samples of gases for CO₂ in rapid succession with an Orsat apparatus. The gas samples for the CO₂ recorders or the Orsat apparatus were drawn at section G (see fig. 2) of the combustion chamber, where combustion was practically complete. Gas pressures in the furnace were controlled by regulating the speed of the exhaust fan. The additional air over the fuel bed was regulated partly by the small fan supplying air to the tuyères and partly by the gas pressure (draft) over the fuel bed. When more air was needed the pressure drop through the tuyères was made greater. With very high rates of combustion and high air supply not enough air could be added over the fuel bed through the tuyères. In such instances some air was admitted over the fuel bed through the fire door, which was left partly open, and through the coal-feeding openings by keeping the coal somewhat low in the magazines. The admission of air through the fire door and through the coal magazines was unsatisfactory and much trouble was experienced in keeping the air supply constant. This method of air introduction also changed to some extent the way of mixing the gases, which change resulted in some inconsistencies in the results.

WEIGHT OF ASH AND REFUSE.

At the end of each test the fire was cleaned, the clinker and the ash accumulated in the ash pit was removed and weighed, and a sample sent to the chemical laboratory for the determination of combustible. In determining the rate of combustion the amount of

^a For description of CO₂ recorders, see Barkley, J. F., and Flagg, S. B., Instruments for recording carbon dioxide in flue gases: Bull. 91, Bureau of Mines, 1915, 60 pp.

this combustible was subtracted from the amount of combustible fired.

SAMPLING AND ANALYZING FURNACE GASES.

As the study of the process of combustion was based entirely on the composition of the gases at various sections along the path of the gases, the sampling of the gases was the most important part of the observations. The sampling of gases in the furnace is a difficult problem, because the temperature ranges from $2,000^{\circ}$ to $3,200^{\circ}$ F., and consequently the samples had to be collected with water-cooled tubes. Also the fact that the gases in the furnace at any one section are not of uniform composition necessitated taking several samples at each cross section and averaging their composition. After a number of preliminary tests it was decided to take 6 to 9 samples at each section, so that 35 to 50 samples had to be collected simultaneously. The distribution of the samples at the various cross sections is shown in figures 4, 5, and 6.

The rate of combustion and the air supply was never under perfect control, so that the composition of the gases at any one point in the furnace varied somewhat in response to the changing condition of the fire. To reduce the effect of such fluctuations the samples were collected over comparatively short periods, ranging from 8 to 40 minutes, and never over the entire test. On some of the tests duplicate samples were collected.

The gas samples were collected at eight cross sections of the combustion chamber, which were designated A to H (see fig. 2), A representing the first vertical section about 5 feet from the center of the grate. On the last series of tests with Pittsburgh coal screenings,

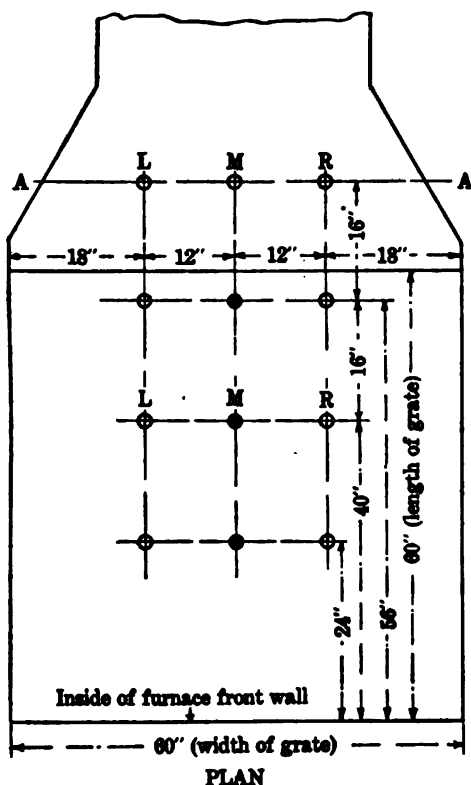


FIGURE 4.—Plan view showing position of gas and tar samplers at surface of the fuel bed and 11 inches above it. Tar-and-gas samplers are indicated by circles, and gas samplers only by dots. L, M, and R indicate left, middle, and right sampler, respectively. A indicates section A of combustion furnace.

samples of gas were also collected at three horizontal sections immediately over the grate, one being at the surface of the fuel bed, one 11 inches from the surface of the fuel bed, and the third 22 inches from the surface of the fuel bed. There were only three samplers

22 inches from the surface of the fuel bed. The position of these sections is shown in figures 4 and 7.

The sampling tubes were inserted through the vertical holes in the arch of the furnace. Two or three tubes were mounted in one water jacket, the intakes of the tubes extending to different depths in the furnace. By this construction two or three gas

samples could be taken through each hole in the roof, and the number of rubber-hose connections for the cooling water was greatly reduced.

GAS-SAMPLING TUBES.

The construction of a triple gas-sampling tube is shown in figure 8. The inlet of the sampler is connected with a half-inch hose to a water supply having a pressure of not less than 24 pounds per square inch. The discharge end of the sampler is connected in a similar way to a drain. Both hose connections are of sufficient length to permit free insertion of the sampler into the furnace or its removal while the water circulates through it. Directions for making these samplers follow:

All joints that come in contact with flames are brazed; joints that remain outside of the furnace are soldered. When the sampler is being brazed or soldered, it should be held in such a position that the molten spelter or solder runs into the joint; this is important if good joints are to be obtained. When the joints to be brazed are being fitted, care should be taken to have the end of the copper tubing extend out far enough so that the spelter will not run down into it. After the joint has been brazed, the protruding end of the copper tube can be filed off. The

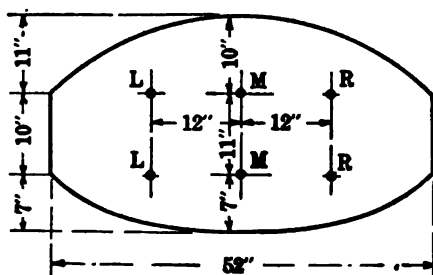


FIGURE 5.—Position of tar and gas samplers at section A of combustion furnace. L, M, and R indicate left, middle, and right sampler, respectively.

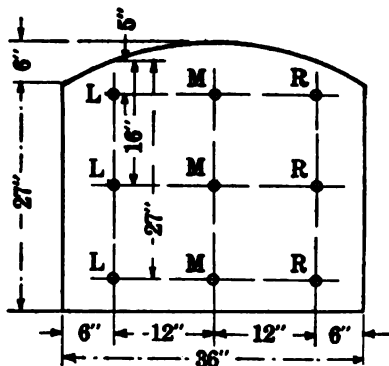


FIGURE 6.—Position of gas samplers at sections beyond section A of combustion furnace. L, M, and R indicate left, middle, and right sampler, respectively.

side outlets on the gas tubes are first fitted, as shown, and then soldered by placing a piece of solder into the side outlet and holding the joints over a gas-torch flame, so that the molten solder runs all around the joint. The hole between the side outlet and the gas tube is drilled after the joint has been soldered. The gas sample is drawn through the side outlet; the opening at the end of the gas tube is closed with a piece of rubber tubing and a glass plug, and is used only for cleaning the tube when stopped with soot.

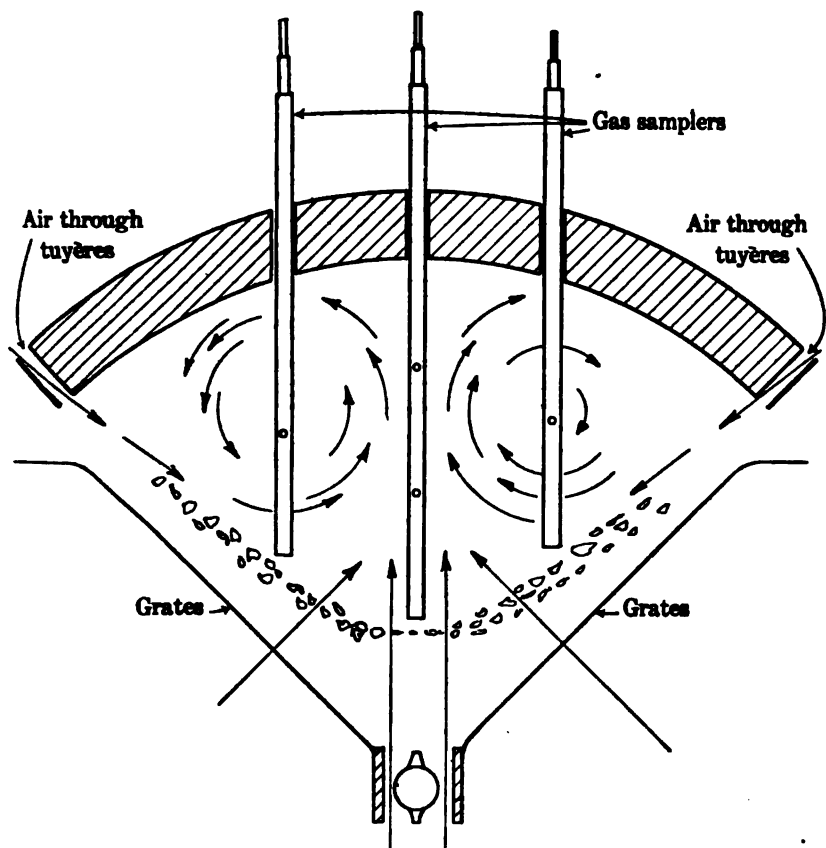


FIGURE 7.—Position of tar and gas samplers over the fuel bed. Circles indicate inlets of samplers. The two side rows took tar and soot samples; the central row, gas only. Entrance of air through tuyères and the whirling motion of the gases that is caused by it are indicated.

GAS-COLLECTING APPARATUS.

The arrangement of the apparatus for collecting gas samples is shown diagrammatically in figure 9. The apparatus consists mainly of the previously described water-cooled sampling tube, a glass gas container, a wash bottle, and a suction pipe leading to a steam ejector. By means of the suction pipe one steam ejector serves to

collect a large number of samples. These various parts of the apparatus are connected with $\frac{1}{4}$ -inch lead pipe, $\frac{1}{8}$ -inch copper tubing, and capillary glass tubing with rubber tubing for joints. The operation of the apparatus is as follows:

The steam ejector maintains a reduced pressure in the suction pipe, thereby causing a continuous current of gas to flow through the apparatus as indicated by the arrows in figure 9. The wash bottle is partly filled with water. The copper tubing leading to the bottle extends below the surface of the water so that as the gas flows through the system the flow makes itself visible by the gas bubbling through the water. Thus the apparatus can not become clogged without the operator knowing it. For a better control of

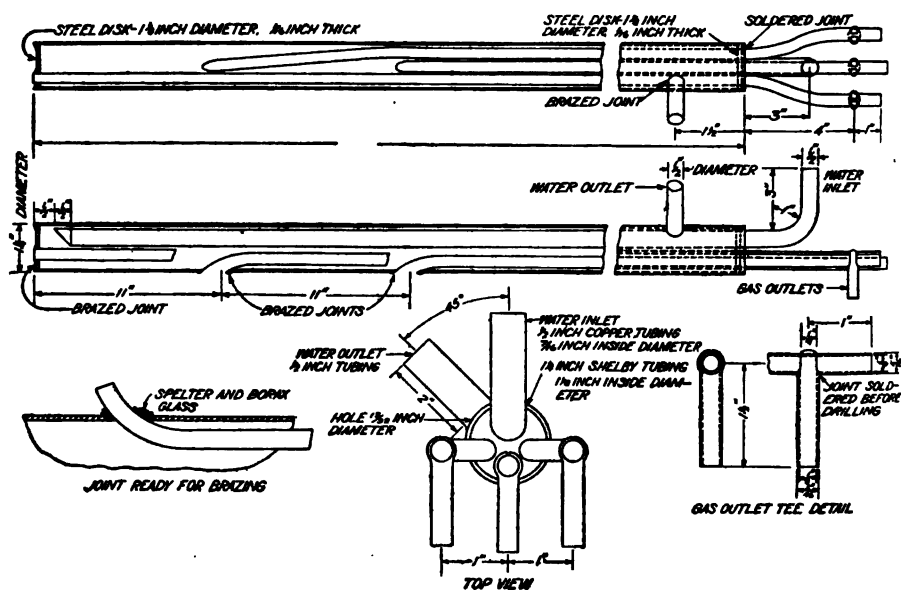


FIGURE 8.—Water-cooled gas sampler having three gas-sampling tubes in one water jacket.

the flow of gas the suction pipe is equipped with a manometer. Usually a pressure of 2 inches of mercury less than the atmospheric pressure is maintained in the suction pipe.

Before a sample is collected for analysis the gas container is filled completely with mercury. When connecting the container to the apparatus the air from the capillary tube is removed by applying strong suction at joint A (see fig. 9). Then the rubber joint is held tightly closed between the thumb and fingers and slipped over the top end of the container. Gas is drawn into the container by allowing the mercury to run out through the lower valve into a glass bottle. The length of the period of collecting gas depends on the rate at which the mercury is allowed to run out of the container. This rate

can be controlled by adjusting the lower valve or by attaching to it a special glass tip having the proper size of opening, which is determined by trial. The inlet of the gas container extends within about one-half inch of the bottom so that as long as the inlet remains below the surface of the mercury the mercury flows out under constant head, being replaced with gas at a uniform rate. The flow of mercury is stopped before its level falls below the inlet.

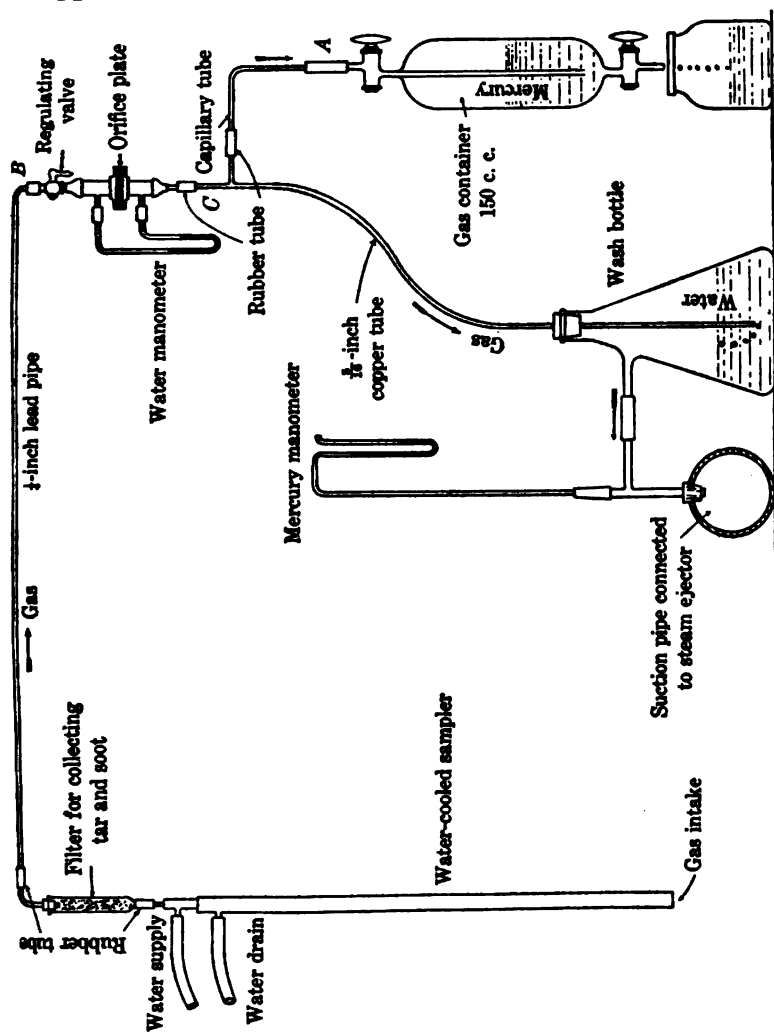


FIGURE 9.—Diagram of apparatus used in sampling gases, tar, and soot. When only gases were sampled the orifice and the filter part were removed, and the end B of the lead pipe was connected directly to the end C of the copper tube.

The gas container also serves as a safe and convenient holder for storing the samples. The mercury forms a seal between the gas and the stopcocks as long as its level does not fall below the opening of the inlet. The seal prevents leakage through the stopcocks. All danger of leakage into the container is entirely eliminated if the gas

sample is placed under slight pressure by forcing in some mercury through the gas inlet tube.

For convenience and safety in handling, three gas containers are mounted together in one portable wooden stand.

APPARATUS FOR COLLECTING SAMPLES OF SOOT AND TAR.

In the last series of tests with Pittsburgh coal screenings a number of samples of soot and tar were collected near the fuel bed with the samples of gases. Of the soot and tar samples six were collected at the surface of the fuel bed, six samples 11 inches from the fuel bed, and six at section A (see fig. 2) of the combustion chamber. The location of the soot and tar samplers is shown in figures 4, 5, and 7. The sampling tubes used for this purpose were somewhat larger than those used for sampling gas alone. The inner metal tube was three-eighths inch in external diameter. Into this tube fitted snugly a removable glass tube which extended to within three-fourths of an inch of the furnace end of the metal tube and protruded about 2 inches from the outside end, forming a lining which could be pulled out and cleaned. The outside end of the glass tube was connected to a filter consisting of a glass tube 12 inches long and three-fourths inch in diameter, filled with asbestos fiber which had been previously burned to remove all combustible material. The fiber was packed around a brass-wire spring to prevent its being drawn by the moving gases to one end of the filter tube and stopping it. No fiber was placed in the long glass tube forming the lining for the sampling tube. The soot and tar was collected partly in the long tube and partly in the filter. In order to obtain quantitative data the volume of gas drawn through the sampler and the filter was measured with small orifices placed in the pipe connection as shown in figure 9. The orifices were calibrated with a small experimental gas meter. The pressure drop through the orifices was measured with U-tube manometers connected as shown in the figure. The soot and tar samples were collected for a period of 20 to 30 minutes; the pressure drop through the orifices was read every five minutes. The readings were plotted with time as abscissa and the weight of gas was computed for small sections of the curve connecting the plotting points. The weight of the gases as thus determined is probably correct within 25 per cent.

After a sample of soot and tar had been collected the long glass tube was pulled out of the sampler, and the deposit carefully washed out with benzol. This deposit, together with that collected in the filter, was taken as the sample of tar and soot.

ANALYSIS OF GAS SAMPLES.

The analyses of gas samples were made with modified Hempel apparatus. Mercury was used in the measuring burette, which was water-jacketed and provided with a Pettersson pressure and temperature compensating tube. Absorption pipettes of the Lindstrom type, fitted with a two-way stopcock and small cup for containing water, were used. By the use of pipettes of this type the error due to the volume of gas in capillary connections is eliminated.

The samples of gas were analyzed for CO_2 , O_2 , CO , CH_4 , H_2 , and unsaturated hydrocarbons. Phosphorus was used for the determination of O_2 . CO was determined by absorption in ammoniacal cuprous chloride solution; three pipettes were used in series, care being taken that the third one always contained a fresh solution. The determinations of CH_4 and H_2 were made over mercury by the slow combustion method with heated platinum coil. The content of unsaturated hydrocarbons in the samples collected over the fuel bed and at section A was determined by the method of absorption in fuming sulphuric acid.

During the early part of this work a special burette, designed for exact analysis and described in Technical Paper 63,^a was used for the determinations of CH_4 and H_2 . Because of the difficulties of obtaining samples with sufficient accuracy to justify exact gas analysis, the use of this burette was discontinued. The determination of CO_2 , O_2 , unsaturated hydrocarbons, and small quantities of CO is within plus or minus 0.1 per cent of the volume of the sample. With large amounts of CO requiring more than one pipette for absorption, and with CH_4 and H_2 , the error may reach at times plus or minus 0.2 per cent.

DETERMINATION OF TAR AND SOOT.

Tar was separated from soot by extraction with hot benzol in Soxhlet extraction apparatus. The soluble part was considered tar and the insoluble combustible part, soot.

The tar and soot deposited in the glass sampling tube and filter were removed by cleaning thoroughly with asbestos fiber and benzol. The asbestos fiber from the filter and that used in cleaning, which together contained all the tar and soot, were placed in an alundum thimble and extracted on a sand bath until the benzol was colorless. The tar, contained in the extraction flask, was freed from benzol by passing dry air at room temperature through the flask until the loss was less than 1 milligram on successive weighings one-half hour apart. The difference between the weight of the flask with tar and that of the clean, dry flask is the weight of tar.

^a Clement, J. K., Fraser, J. C. W., and Augustine, C. E., Factors governing the combustion of coal in boiler furnaces: Tech. Paper 63, Bureau of Mines, 1914, p. 17.

After extraction was completed the alundum thimble containing the soot mixed with asbestos fiber was dried to constant weight at 105° to 110° C., and the soot burned off in a muffle furnace at red heat. The loss in weight of the dried thimble due to burning is the amount of soot.

The method does not yield strictly accurate results for two reasons. First, benzol combines to some extent with tar yielding insoluble compounds, thereby giving low results for tar and high results for soot.^a The error from this source is not over 5 per cent of the weight of the tar. Second, while dry air is being passed at atmospheric temperature through the flask to remove benzol, some of the light oils contained in the tar are driven off with the benzol and lost. The error from this source is not greater than 2 per cent.

TEMPERATURE MEASUREMENTS.

Observations of the temperature within the combustion chamber were made at 20-minute intervals in all the tests. The temperatures

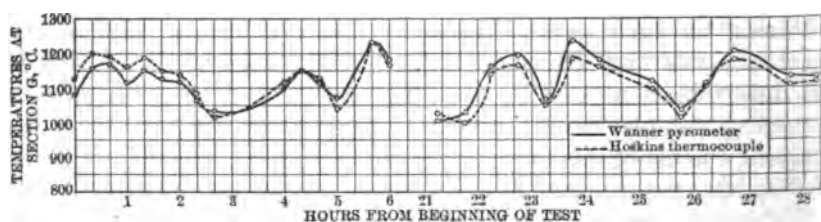


FIGURE 10.—Comparison of temperatures in combustion chamber at section G, as measured with Hoskins thermocouple and Wanner pyrometer.

were measured with a Wanner optical pyrometer through the side holes at sections A and G (see fig. 2) of the furnace and in many tests also at C and E. On account of the high temperature and the presence of slag in the gases it was not practicable to use thermocouples.

The temperature indicated by the optical pyrometer is the temperature of the inner surface of the opposite wall when there is no flame. When the chamber is filled with flame, the pyrometer measures the temperature of the flame. To determine the relation between the temperature of the wall as indicated by the optical pyrometer and the temperature indicated by a thermocouple suspended in the path of gases, a Hoskins thermocouple was placed at section G with its hot junction in the center of the section, and simultaneous readings were made with the couple and the optical pyrometer. The results are given in figure 10. As shown in the figure the two methods of temperature measurements agree within about 25° C. This relation

^a Weiss, J. M., Free carbon; its nature and determination in tar products: Jour. Ind. and Eng. Chem., vol. 6, 1914, p. 279.

was determined only for readings that were taken when there was no flame present and the gases were transparent. Usually the gases leaving the fuel bed carry with them in suspension soot, tar, and a small amount of ash. When present in large quantities these substances form an opaque cloud of flame which obscures the wall beyond, and as the individual particles are incandescent, it is the temperature of the cloud or flame that is measured by the optical pyrometer and not the temperature of the gases or of the wall. As the particles of soot are hot from their own combustion, besides receiving heat from the surrounding gases, it is by no means certain that the temperature measured by the optical pyrometer is the average temperature of the gases and flame passing through the combustion chamber.

GAS-PRESSURE MEASUREMENTS.

Readings of gas pressures were taken at 20-minute intervals on all the tests. The pressures recorded were those in the ash pit and at sections A and G of the furnace. Ellison inclined-tube draft gages were used for this purpose.

COALS USED IN THE TESTS.

As previously stated, three kinds of coal were used in the tests, namely, Pocahontas, Pittsburgh, and Illinois coal.

The Pocahontas coal used was from the Norfolk mine of the Pocahontas Consolidated Coal Co., at Vivian, W. Va., which works the Pocahontas No. 3 bed. The average analysis of two car loads of the coal is given in column 2, Table 1 (p. 24). The coal was screened at the mine by passing it over a $\frac{3}{4}$ -inch and through a $1\frac{1}{2}$ -inch bar screen. In the necessary handling the size of the coal was somewhat reduced. Actual sizing tests with wire screens showed the sizes given in column 2, Table 2.

The Pittsburgh coal used came from the No. 4 mine of the Jamison Coal & Coke Co., situated 4 miles north of Greensburg, Westmoreland County, Pa. The average analysis of the car samples is given in column 3, Table 1. The coal was shipped as run-of-mine and had to be broken to the desired size before using. Column 3, Table 2, gives the percentage of the different sizes of the coal as burned. The sizes were determined by running over shaking screens samples of coal from several tests.

The Illinois coal came from bed No. 6, and was mined in the Vermilion mine of the Bunsen Coal Co., at Georgetown, Vermilion County. The analysis of the coal is given in column 4, Table 1. The coal was shipped as run-of-mine and was broken with a hammer before using it. The sizes of the coal as it was burned are given in column 4, Table 2.

In the last series of 23 tests, Pittsburgh coal screenings similar in composition to the Pittsburgh run-of-mine coal (column 3, Table 1) were used. These screenings, used as regular fuel for the power plant of the experiment station, varied greatly in size, the largest pieces being about 2 inches across. On the average about one-third of the coal would run through a $\frac{1}{4}$ -inch screen. This series of tests was undertaken with the object of studying the combustion between the surface of the fuel bed and section A of the furnace. The greater part of the combustible rising from the fuel bed is burned in this space, as shown in figure 13.

TABLE 1.—*Analysis of coals used in the tests.*

PROXIMATE ANALYSIS OF COAL AS RECEIVED.

Constituent.	Pocahontas coal.	Pittsburgh coal.	Illinois coal.
Moisture.....per cent..	2.21	2.51	16.16
Volatile matter.....do..	15.78	30.28	34.09
Fixed carbon.....do..	71.65	56.82	39.19
Ash.....do..	10.36	10.39	10.56
	100.00	100.00	100.00

ULTIMATE ANALYSIS OF DRY COAL.

Hydrogen.....per cent..	3.92	4.82	4.66
Carbon.....do..	80.90	76.57	69.63
Nitrogen.....do..	1.06	1.55	1.49
Oxygen.....do..	2.97	4.99	9.85
Sulphur.....do..	.56	1.41	2.08
Ash.....do..	10.69	10.66	12.39
	100.00	100.00	100.00
Calorific value per pound, as received.....B. t. u..	13,762	13,365	10,433

TABLE 2.—*Sizes of coal as burned.*

[Square holes in screen.]

Size of coal.	Per cent by weight.		
	Pocahontas.	Pittsburgh.	Illinois.
Through $\frac{1}{4}$ -inch mesh.....	5	10	16
Over $\frac{1}{4}$ -inch and through $\frac{1}{2}$ -inch mesh.....	5	8	12
Over $\frac{1}{2}$ -inch and through $\frac{3}{4}$ -inch mesh.....	6	14	21
Over $\frac{3}{4}$ -inch and through 1-inch mesh.....	62	23	27
Over 1-inch and through $1\frac{1}{4}$ -inch mesh.....	22	45	24
	100	100	100

METHOD OF SAMPLING.

A car sample was taken at the time of unloading the cars and sent to the chemical laboratory for proximate and ultimate analysis and determination of heat units. During each test a sample of about

100 pounds was collected and a part of it submitted for proximate analysis. A sample of ash and refuse was collected at the close of each test for the determination of the percentage of combustible.

RESULTS OF THE EXPERIMENTS.

In all, 100 tests with complete sets of observations were made. Of these, 23 tests were with Pocahontas run-of-mine coal, 37 with Pittsburgh run-of-mine coal, 17 with Illinois run-of-mine coal, and 23 with Pittsburgh screenings. With each coal, tests were run at rates of combustion of 20, 30, 40, 50, and 60 pounds of coal per square foot of grate per hour. A series of three or more tests was run with each rate of combustion, the tests in any one of these series being run with different air supply. Thus there were investigated three factors—the kind of coal, the rate of combustion, and the air supply.

VARIATION OF CONDITIONS DURING TESTS.

Notwithstanding the provisions made in designing the apparatus and the care taken in its operation, conditions varied considerably during the tests. These variations are shown graphically in figures 11 and 12. Perhaps the most serious variation affecting the results of the tests is that of the percentage of CO₂ in the furnace gases. This variation seems inherent with the stoker, as the coal on the upper part of the stoker caked instead of sliding down uniformly. On account of this caking, bare spots frequently appeared below the crusted area; the crust had to be broken and the holes covered with coal brought down with a hand tool. These holes and the work of covering them caused irregular air supply and resulting variation of CO₂ in the gases. The change in temperature along the chamber is due to heat absorption by the gas samplers and other causes explained in the discussion of temperature variation in pages 109 to 111.

Experiments made on a large scale approximating commercial conditions are subject to variations that do not occur in small laboratory apparatus in which the conditions of operation can be more accurately controlled. Tests with small laboratory apparatus have the drawback that one is never sure to what extent the results obtained are applicable to commercial types of furnaces. In choosing apparatus for experiment one has the alternative of equipment of nearly commercial type and size with large variation of operating conditions, or a small furnace with controllable conditions giving accurate results, which, however, may not be applicable to furnaces of commercial size.

TABULATED RESULTS.

The results of the test are given in Table 3 and are arranged as follows: (1) The tests are grouped according to the kind of coal used; (2) tests with the same kind of coal are grouped according to the rate

of combustion; and (3) the tests with nearly the same rate of combustion are arranged according to the air supply.

On tests 104 to 113, inclusive, of the series with Pocahontas coal, duplicate sets of gas samples were collected. The sets are denoted as

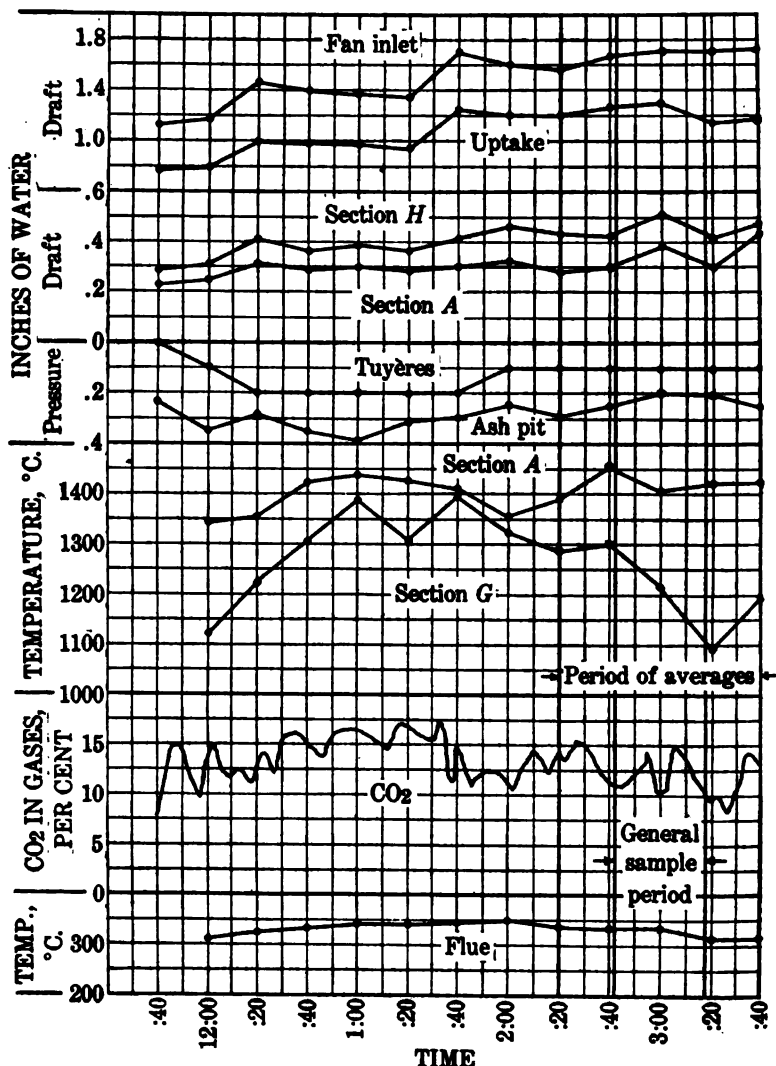


FIGURE 11.—Variation of conditions and period of sampling in test 179.

104-1, 104-2, 105-1, 105-2, etc. In tests with the Pittsburgh screenings, 12 to 21 gas samples were taken at sections in front of section A of the combustion chamber. At these sections and at section A were collected also 18 soot and tar samples.

In Table 3 the tests are grouped according to the kind of coal used in the tests. Column 1 gives the serial number of the test and

column 2 the weight of coal fired per square foot of grate per hour. The figures given in this column were obtained by dividing the total weight of coal by the length of test in hours given in column 3, and by 25, which is the projected area of the grate in square feet.

Column 3 gives the time of starting and stopping the test, from which may be obtained the total length of test in hours. This includes the

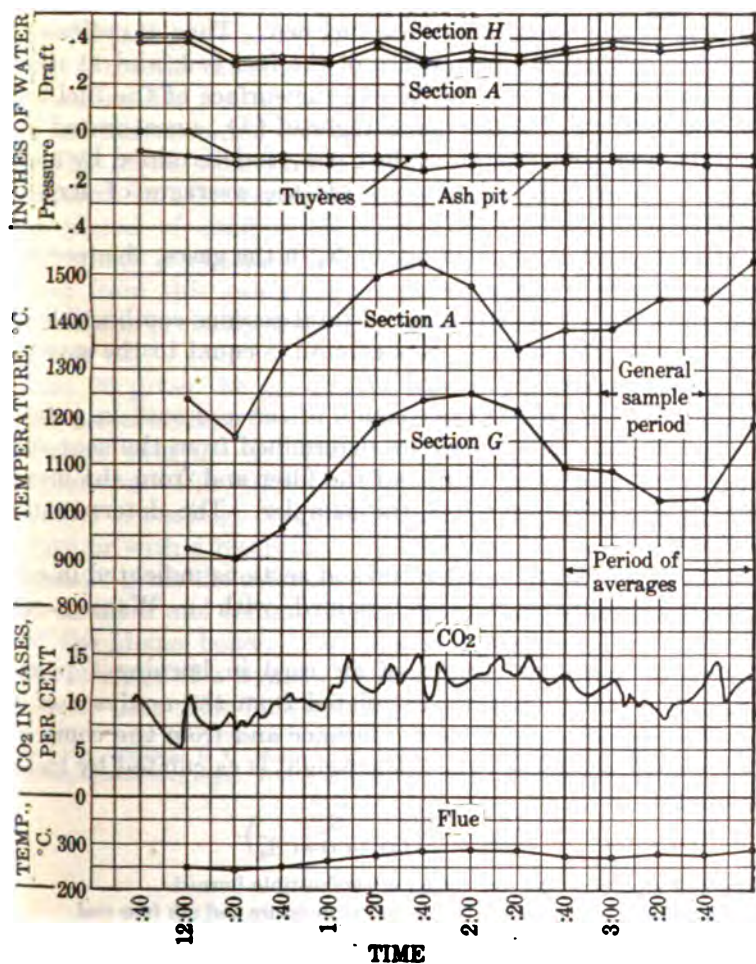


FIGURE 12.—Variations of conditions and period of sampling in test 173.

entire period during which coal was weighed and burned at the desired rate.

Column 4 gives the length of the part of the test used for obtaining the averages of gas pressures and temperatures. This period started 30 to 60 minutes before the collection of gas samples was begun, and ended 30 to 60 minutes after the collection of gas samples was completed.

Column 5 gives the period during which the samples of gas were collected. The three different periods of columns 3, 4, and 5 are indicated in figures 11 and 12.

Column 6 shows in what section of the furnace the gas samples were collected. The situation of the sections is shown in figures 2, 4, 5, 6 and 7. In the series of tests with Pittsburgh screenings the two figures above A, column 6, indicate the distance in inches of the intake of the sampler from the surface of the fuel bed. Thus, 0 indicates that the sampler was just at the surface of the fuel bed, and 11 indicates that the sampler was 11 inches above the surface of the fuel bed.

Columns 7 to 12 give the percentages of CO_2 , unsaturated hydrocarbons, O_2 , CO , CH_4 , and H_2 in the gases, as determined by analysis. The figures given in these columns are the averages of six to nine samples.

Column 13 gives the percentage of N_2 in the gases, this percentage being determined by difference.

Column 14 gives the percentage of total gaseous combustible in the furnace gases. The value in this column is equal to the sum of the values in columns 8, 10, 11, and 12.

Columns 15 and 16 give the weights of tar and soot in grams per cubic foot of gas. The values are determined from the soot and tar collected in the sampling tube and the filter and from the measured volume of gas drawn through the sampler. The determination is correct within about 25 per cent.

Column 17 shows the temperatures at sections indicated in column 6. The temperatures were determined with a Wanner optical pyrometer.

Column 18 gives the weight of air used in burning 1 pound of combustible. The values are calculated from the analyses of gases collected at section G or H of the furnace and from the composition of the combustible in the coal. The weight is calculated by means of the formula:

$$W = 3.03 C \left(\frac{N_2}{\text{CO}_2 + \text{CO} + \text{CH}_4} \right).$$

W = pounds of air supplied per pound of combustible burned.

C = Weight of carbon in pounds per pound of moisture and ash free coal.

N_2 , CO_2 , CO , and CH_4 = percentages in the furnace gases at section G or H.

The amount of combustible burned is equal to the amount fired minus the amount of combustible in the refuse.

Column 19 gives the excess of air used in the combustion, expressed in percentage of the weight of air theoretically needed to completely burn 1 pound of combustible. The excess is equal to the weight of air actually supplied to the furnace minus the weight theoretically needed. The excess air is expressed as a percentage of the weight theoretically required. The weight of air theoretically required is

computed by means of the formula, $A = 34.48H + 11.58C + 4.336(S-O)$, in which H, C, S, and O are the percentages of hydrogen, carbon, sulphur, and oxygen in moisture and ash free coal. The formula is not exact, because (1) it rests on the assumption that the composition of the combustible matter in the refuse is the same as the composition of the moisture and ash free coal fired; (2) no correction is made for the combustible matter that may be carried away with the furnace gases as tar and soot; and (3) no correction is made for the nitrogen in the coal. The assumption that the composition of the combustible matter in the refuse is the same as the composition of the moisture and ash free coal is obviously not correct, the combustible in the refuse being largely carbon with very little volatile matter in it. Consequently the weight of carbon in the furnace gas per pound of combustible burned is somewhat less than the value C, which is obtained from the ultimate analysis of the coal. The error from this assumption regarding the combustible matter in the refuse is probably not over 1 per cent, as was shown in Technical Paper 63.^a

Column 20 gives the pressure of air in the ash pit in inches of water above the atmospheric pressure. Column 21 gives the pressure of gases over the fuel bed in inches of water below the atmospheric pressure. The values in these columns are each the average of a number of readings obtained either with an Ellison inclined tube draft gage or with a recording "hydro" gage.

Column 22 gives the pressure of the gases at section H in the combustion chamber and column 23 the pressure of the gases in the uptake of the Heine boiler. The values given in these columns are each the average of a number of readings obtained with an Ellison draft gage. The table follows.

^a Clement, J. K., Frazer, J. C. W., and Augustine, C. E., Factors governing the combustion of coal in boiler furnaces, a preliminary report: Tech. Paper 63, Bureau of Mines, 1914, p. 30.

TABLE 3.—Results of combustion tests.

POCAHONTAS COAL.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperatures and pressures.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.							Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.	Temperature.	Weight of air per pound of combustible.	Excess of air.	Gas pressure, inches of water.								
						CO ₂	Unsaturated hydrocarbons.	O ₂	CO.	CH ₄	H ₂	N ₂						Total gaseous combustible.	In ash pit, above atmospheric.	Over fuel bed, below atmospheric.	At rear of chamber, below atmospheric.	In uptake, below atmospheric.				
10-2	20.6	11 a. m. to 3.40 p. m.	1.40 to 3.40 p. m.	3.04 to 3.10 p. m.	A B C D E F G H	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Gms.	° C.	Lbs.	P. ct.	0.06	0.08	0.11						
						15.1	2.4	4.5	1.6	0.0	0.0	0.0	0.0	1.583	1,583											
						15.1	3.6	1.2	1.2	0.0	0.0	0.0	0.0													
						13.5	6.2	0.0	0.0	0.0	0.0	0.0	0.0													
						16.7	1.8	0.0	0.0	0.0	0.0	0.0	0.0													
10-1	20.6	11 a. m. to 3.40 p. m.	1.40 to 3.40 p. m.	2.14 to 2.19 p. m.	A B C D E F G H	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Gms.	° C.	Lbs.	P. ct.	0.06	0.08	0.11						
						15.1	4.1	1.6	1.6	0.0	0.0	0.0	0.0	1,583	1,583											
						15.1	4.5	2.0	2.0	0.0	0.0	0.0	0.0													
						14.0	5.2	0.0	0.0	0.0	0.0	0.0	0.0													
						16.3	2.7	0.0	0.0	0.0	0.0	0.0	0.0													
108-2	22.2	8.48 a. m. to 3.50 p. m.	1.40 to 3.40 p. m.	3.13 to 3.20 p. m.	A B C D E F G H	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Gms.	° C.	Lbs.	P. ct.	0.04	0.11	0.14						
						14.1	3.7	2.4	2.4	0.0	0.0	0.0	0.0	1,540	1,540											
						15.2	3.0	0.0	0.0	0.0	0.0	0.0	0.0													
						14.6	3.3	0.0	0.0	0.0	0.0	0.0	0.0													
						15.0	2.6	0.0	0.0	0.0	0.0	0.0	0.0													

100-1	22.2	8.48 a. m. to 3.50 p. m.	1.40 to 3.40 p. m.	2.07 to 2.13 p. m.	A 13.4 B 14.1 C 13.2 D 14.1 E 14.2 F 14.4 G 14.1 H 12.9	6.0 5.5 5.5 5.2 5.3 5.0 5.3 0	7	1,140	.04	.11	.14
100-2	20.0	10 a. m. to 3 p. m.	1.20 to 3 p. m.	2.12 to 2.18 p. m.	A 7.3 B 6.9 C 8.6 D 8.6 E 8.4 F 8.5 G 8.6 H 8.4	10.2 12.4 11.0 10.8 11.0 11.1 11.0 0	3.5 3 0 0 0 0 0 0	1,342	0	.11	.15
100-1	20.0	10 a. m. to 3 p. m.	1.20 to 3 p. m.	1.50 to 1.55 p. m.	A 8.2 B 8.8 C 7.5 D 9.1 E 8.3 F 8.1 G 7.7 H	10.6 11.8 10.8 11.3 11.2 11.7 0	8 3 0 0 0 0 0	1,344	0	.15	.11
100-1	28.0	11 a. m. to 4.40 p. m.	2.40 to 4.20 p. m.	3.05 to 3.10 p. m.	A 11.6 B 15.0 C 16.4 D 16.4 E 16.0 F 16.8 G 16.3 H 14.9	2.2 2.6 1.9 2.6 2.8 2.6 2.9 0	8.4 1.0 2 0 0 0 0 0	1,625	.15	.11	.15
100-2	28.0	11 a. m. to 4.40 p. m.	2.40 to 4.20 p. m.	3.57 to 4.03 p. m.	A 12.9 B 15.3 C 15.1 D 15.2 E 15.6 F 15.4 G 14.2 H	2.2 2.3 4.0 3.7 3.3 2.8 0	5.0 1 0 0 0 0 0	1,625	.15	.11	.15
100-2	28.0	11 a. m. to 4 p. m.	1.20 to 4 p. m.	3.25 to 3.32 p. m.	A 12.0 B 15.1 C 13.9 D 15.4 E 15.5 F 15.3 G 15.4 H	2.5 2.7 6.2 3.9 3.6 3.7 4.0 0	5.7 4 0 0 0 0 0 0	1,594	.11	.10	.15
100-2	28.0	11 a. m. to 4 p. m.	1.20 to 4 p. m.	3.25 to 3.32 p. m.	A 12.0 B 15.1 C 13.9 D 15.4 E 15.5 F 15.3 G 15.4 H	2.5 2.7 6.2 3.9 3.6 3.7 4.0 0	5.7 4 0 0 0 0 0 0	1,594	.11	.10	.15

See figure 2 for diagram of furnace, showing sections A to H.

TABLE 3.—Results of combustion tests—Continued.

POCAHONTAS COAL—Continued.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperatures and pressures.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.										Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.	Temperature.	Weight of air per pound of combustible.	Excess of air.	Gas pressure, inches of water.					
						CO ₂	Unsaturated hydrocarbons.	O ₂	CO.	CH ₄	H ₂	N ₂	Total gaseous combustible.	In ash pit, above atmospheric.	Over fuel bed, below atmospheric.						At rear of chamber, below atmospheric.	In uptake, below atmospheric.				
116	42.8	11 a. m. to 4.20 p. m.	3 to 4.20 p. m.	3.33 to 3.39 p. m.	A B C D E F G H	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Gms.	Gms.	° C.	Lbs.	P. ct.	0.15	0.12	0.18					
						11.2	8.9	7.8	1.9	1.4	1.4	77.2	7.5			1,515										
						11.5	7.8	7.1	1.1	1.1	1.1	76.9	3.5													
						11.4	7.1	7.6	0	0	0	80.9	9													
						11.6	7.6	7.6	0	0	0	80.9	6													
121	61.0	11 a. m. to 3.20 p. m.	12.40 to 3.20 p. m.	2.01 to 2.07 p. m.	A B C D E F G H	10.4	9.0	0	0	0	0	77.2	7.5			1,302	21.3	78.9	.51	.26	.44					
						14.3	1.0	6.0	1.1	1.1	1.1	76.9	3.5			1,618										
						15.8	1.2	2.8	1.3	1.1	1.1	76.9	1.8													
						16.3	2.0	1.3	1.3	1.1	1.1	80.9	9													
						16.9	1.3	1.3	1.3	1.1	1.1	80.9	6													
120	55.7	11 a. m. to 3.40 p. m.	2 to 3.40 p. m.	2.40 to 2.56 p. m.	A B C D E F G H	17.5	1.0	1.1	1.1	1.1	1.1	81.1	5			1,475	12.9	7.3	.43	.32	.47					
						17.6	1.8	2.2	1.1	1.1	1.1	81.1	3													
						17.2	1.4	1.1	1.1	1.1	1.1	77.0	6.8			1,549										
						13.7	2.5	2.5	2.1	2.1	2.1	80.4	1.1													
						15.9	1.8	1.9	1.9	1.1	1.1	81.5	2.4													

PITTSBURGH RUN-OF-MINE.

124	38.9	11 a. m. to 3.30 p. m.	2 to 3.20 p. m.	2.34 to 2.41 p. m.	A	12.3	6.3	2.0	.2	79.0	2.4	1,560	.45	.20	.39
					B	12.1	6.3	.1	.0	80.5	.1				
					C	12.8	6.6	.0	.0	80.6	.0				
					D										
					E										
					F										
					G							1,318			
					H	11.1	8.4	.0	.0	80.5	.0		20.0	66.3	
123	38.2	11 a. m. to 3.40 p. m.	1.20 to 3.40 p. m.	2.25 to 2.32 p. m.	A	13.0	5.7	1.9	.1	79.1	2.2	1,523	.49	.28	.38
					B	13.0	6.8	.0	.2	80.0	.2				
					C	11.4	8.2	.0	.0	80.4	.0				
					D										
					E										
					F										
					G							1,244			
					H	10.4	9.4	.0	.0	80.2	.0		21.3	76.8	
125	66.5	11.40 a. m. to 3.20 p. m.	2.20 to 3.20 p. m.	2.45 to 2.53 p. m.	A	10.6	3.8	8.3	.2	75.8	9.8	1,599	.55	.30	.50
					B	14.9	2.0	3.3	.2	79.2	3.9				
					C	14.3	3.9	1.1	.3	80.3	1.4				
					D	15.9	2.0	1.0	.1	80.7	1.4				
					E	15.3	2.8	.7	.0	81.5	.9				
					F	16.4	2.0	.4	.0	81.0	.6				
					G	13.5	5.4	.4	.1	80.6	.5	1,386			
					H	16.5	2.3	.3	.0	80.8	.4		13.2	10.1	

160	22.3	11 a. m. to 3.40 p. m.	1.40 to 3.40 p. m.	2.09 to 3.19 p. m.	A	12.4	2.6	6.0	.0	76.4	7.6	1,554	.06	.09	.10
					B	14.1	2.3	2.6	.0	79.7	2.9				
					C	15.0	2.5	1.6	.0	80.0	2.5				
					D	14.8	2.0	1.1	.0	80.5	1.7				
					E	14.9	2.6	1.5	.0	80.2	2.3				
					F	14.9	3.1	1.0	.0	80.3	1.7				
					G	13.6	4.8	1.2	.0	79.8	1.8	1,332	13.2	14.0	
					H	15.2	3.1	.7	.0	80.7	1.0				
155	19.6	11 a. m. to 2.40 p. m.	12.40 to 2.40 p. m.	1.41 to 1.48 p. m.	A	11.1	2.4	10.4	.0	72.3	13.2	1,079	.07	.10	.11
					B	14.6	2.3	1.7	.0	80.0	2.1				
					C	14.9	2.5	.6	.0	80.8	.8				
					D	15.1	2.6	.1	.0	81.2	.1				
					E	14.5	3.9	.0	.0	81.6	.0				
					F	15.4	3.3	.0	.0	81.3	.0	1,433			
					G	15.2	2.6	.0	.0	81.2	.0		13.5	16.3	
					H	15.6	2.3	.0	.0	81.1	.0				

127	31.0	11.30 a. m. to 3.40 p. m.	2.20 to 3.40 p. m.	3.01 to 3.08 p. m.	A 13.8 B 12.3 C 12.6 D 10.3 E 10.4 F 16.1 G 16.6 H 15.6	3.3 2.5 2.6 2.4 3.0 2.2 2.3	3.5 2.9 3.0 1.0 1.0 1.0 1.0	1.1 2.0 2.4 3.0 0.0 0.0 0.0	77.7 80.4 81.0 80.9 80.6 80.9 80.9	2.0 2.0 1.2 1.0 1.0 1.0 1.0	1.478	13.5 16.1	.09	.19	.23	.25	.12	.18
147	30.0	11 a. m. to 2.40 p. m.	1.40 to 2.40 p. m.	2.11 to 2.18 p. m.	A 11.1 B 12.1 C 12.6 D 12.3 E 12.4 F 12.4 G 12.7 H 11.9	6.4 6.4 6.2 6.6 6.6 6.0 6.4 6.9	2.3 2.5 3.0 2.1 2.0 2.0 2.0 2.0	0.9 0.2 0.1 0.0 0.0 0.0 0.0 0.0	79.3 80.8 81.0 81.3 81.0 81.0 81.2 81.2	3.2 2.7 1.0 1.1 1.0 1.0 1.0 1.0	1.562	17.7 32.6	.09	.19	.23	.25	.12	.18
128	29.0	10.30 a. m. to 3.20 p. m.	2.20 to 3.20 p. m.	2.47 to 2.54 p. m.	A 11.8 B 12.3 C 11.4 D 11.8 E 11.8 F 11.8 G 10.6 H 10.6	7.5 6.5 7.6 7.4 7.4 7.4 7.4 8.6	1.0 1.3 1.0 1.0 1.0 1.0 1.0 1.0	0.0 0.0 0.0 0.0 0.0 0.0 0.0 0.0	80.6 80.9 80.9 80.8 80.8 80.8 80.8 80.8	1.1 1.1 1.1 1.1 1.1 1.1 1.1 1.1	1.118	19.9 70.8	.19	.18	.26	.25	.12	.18
128	20.8	10.30 a. m. to 3.20 p. m.	2 to 3.20 p. m.	2.37 to 2.44 p. m.	A 10.5 B 11.5 C 11.1 D 10.3 E 11.1 F 11.8 G 11.3 H 9.2	7.0 7.6 8.0 9.0 7.9 7.4 8.0 10.4	2.6 2.7 1.0 0.0 1.0 0.0 0.0 0.0	0.9 0.9 0.9 0.9 0.9 0.9 0.9 0.9	77.9 80.8 80.8 80.9 80.9 80.7 80.4 80.4	4.6 1.1 1.1 1.1 1.1 1.1 1.1 1.1	1.434	22.8 96.8	.11	.15	.21	.25	.12	.18
162	29.6	11.20 a. m. to 3.40 p. m.	2.40 to 3.40 p. m.	2.44 to 3.28 p. m.	A 10.4 B 9.4 C 8.8 D 8.8 E 9.1 F 8.7 G 8.8 H 10.4	7.6 9.7 10.5 10.4 10.2 10.4 10.6 10.6	3.4 3.4 1.0 0.0 0.0 0.0 0.0 0.0	1.5 1.5 0.0 0.0 0.0 0.0 0.0 0.0	77.1 81.4 80.6 80.7 80.9 80.7 80.8 80.6	4.9 1.1 1.1 1.1 1.1 1.1 1.1 1.1	1.983	22.9 106.8	.05	.20	.23	.25	.12	.18
130	39.1	10.20 a. m. to 3 p. m.	2 to 3 p. m.	2.26 to 2.33 p. m.	A 14.6 B 13.7 C 13.7 D 13.7 E 13.7 F 13.7 G 13.6 H 15.4	2.3 2.1 2.1 2.2 2.2 2.2 2.2 2.4	4.2 1.3 1.3 1.3 1.3 1.3 1.3 1.3	0.8 0.8 0.8 0.8 0.8 0.8 0.8 0.8	78.0 78.3 78.3 80.7 80.7 80.7 81.0 81.0	5.1 2.7 2.7 1.2 1.2 1.2 1.2 1.2	1.986	13.5 16.6	.24	.15	.28	.25	.12	.18

136	40.0	11.20 a. m. to 3.20 p. m.	1.20 to 3.20 p. m.	1.44 to 2.59 p. m.	A 10.9 B 12.1 C 12.4 D 12.1 E 12.5 F 12.4 G 12.2 H 12.6	5.9 5.8 5.3 6.7 6.0 6.4 6.7 6.4	5.0 8 5 2 4 2 1 1	0 0 0 0 0 0 0 0	1.2 77.1 30.3 80.7 81.0 81.0 81.0 80.9	6.2 1.1 1.1 3 5 2 1 1	1,146	.09	.33	.38
132	38.7	10.30 a. m. to 3.40 p. m.	2.20 to 3.40 p. m.	2.58 to 3.05 p. m.	A 12.0 B 13.7 C 13.6 D 13.0 E 13.2 F 11.8 G 12.9 H 12.0	4.7 4.6 5.0 5.9 5.4 5.6 5.5 6.4	4.1 8 1 0 0 0 2 0	.2 1.4 5 0 0 0 0 0	77.6 80.3 81.3 81.1 81.4 82.5 81.4 81.0	5.7 1.4 1 0 0 1 2 0	1,135	.28	.19	.27
135	44.9	10.30 a. m. to 3.40 p. m.	2.20 to 3.40 p. m.	2.55 to 3.02 p. m.	A 12.8 B 14.1 C 13.9 D 13.4 E 14.5 F 13.8 G 13.2 H 12.3	5.2 4.6 4.9 5.5 4.4 5.0 5.3 6.7	2.3 4 0 0 0 0 0 0	.1 1 0 0 0 0 0 0	78.9 80.8 81.1 81.1 81.1 81.2 81.5 81.0	3.1 5 1 0 0 0 0 0	1,002	.34	.20	.31
129	39.2	10.30 a. m. to 3.40 p. m.	2 to 3.40 p. m.	2.48 to 2.55 p. m.	A 11.5 B 13.4 C 13.3 D 11.5 E 11.5 F 11.1 G 12.1	5.4 4.8 5.3 5.3 5.3 5.3 5.3 6.8	3.5 7 0 0 0 0 0 0	.9 1 0 0 0 0 0 0	78.7 80.9 81.4 81.4 81.4 81.0 81.0 81.1	4.4 9 0 0 0 0 0 0	1,004	.22	.18	.27
158	40.0	11.20 a. m. to 3.40 p. m.	2 to 3.40 p. m.	2.09 to 3.24 p. m.	A 10.2 B 11.5 C 11.6 D 11.4 E 11.6 F 11.5 G 11.1 H 11.4	5.7 7.1 7.2 7.6 7.2 7.4 8.0 7.7	6.1 5 4 3 3 1 1 1	.0 1.3 0 0 0 0 0 0	76.7 80.8 80.7 80.7 81.1 81.0 80.8 80.8	7.4 6 5 3 1 1 1 1	1,120	.13	.24	.37
131	38.4	8 to 11.20 a. m.	10 to 11.20 a. m.	10.39 to 10.46 a. m.	A 11.7 B 11.9 C 12.0 D 11.0 E 11.9 F 13.0 G 12.8 H 10.3	7.8 6.8 7.0 8.3 7.3 7.2 6.5 9.2	1.6 9 3 0 0 0 0 0	.1 2 1 0 0 0 0 0	78.4 80.1 80.5 80.7 80.8 81.0 80.7 80.5	2.1 1.2 5 0 0 1 0 0	1,416	.16	.26	.35

TABLE 3.—Results of combustion tests—Continued.

PITTSBURGH RUN-OF-MINE—Continued.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperatures and pressures.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.										Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.	Temperature.	Weight of air per pound of combustible.	Excess of air.	Gas pressure, inches of water.			
						CO ₂	Unsat- urated hy- drocarbons.	O ₂	CO.	CH ₄	H ₂	N ₂	Total gaseous combustible.	In ash pit, above atmospheric.	Over fuel bed, be- low atmospheric.						At rear of cham- ber, below at- mospheric.	In uptake, below atmospheric.		
157	37.9	12 m. to 4.20 p. m.	2.20 to 4.20 p. m.	2.49 to 4 p. m.	A B C D E F G H	P. ct. 9.0 10.4 10.4 10.1 10.2 10.0 10.1 10.2	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 9.1 8.4 8.6 8.9 8.9 8.9 8.9 8.9	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 77.5 80.7 80.9 80.9 80.9 80.9 80.9 80.9	P. ct. 4.4 5 5 5 5 5 5 5	Gms. 1,449	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	28	23	23	23	
						P. ct. 10.5 12.1 12.1 12.2 12.2 12.1 12.1 12.1	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 77.5 80.7 80.9 80.9 80.9 80.9 80.9 80.9	P. ct. 4.4 5 5 5 5 5 5 5	Gms. 1,449	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	28	23	23	23	
						P. ct. 10.5 12.1 12.1 12.2 12.2 12.1 12.1 12.1	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 77.5 80.7 80.9 80.9 80.9 80.9 80.9 80.9	P. ct. 4.4 5 5 5 5 5 5 5	Gms. 1,449	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	28	23	23	23	
						P. ct. 10.5 12.1 12.1 12.2 12.2 12.1 12.1 12.1	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 77.5 80.7 80.9 80.9 80.9 80.9 80.9 80.9	P. ct. 4.4 5 5 5 5 5 5 5	Gms. 1,449	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	28	23	23	23	
134	48.0	10.30 a. m. to 3.40 p. m.	1.40 to 3.40 p. m.	2.29 to 2.46 p. m.	A B C D E F G H	P. ct. 10.5 14.1 15.4 16.2 17.1 16.5 16.7 16.0	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 72.7 78.4 76.4 80.6 80.8 80.9 81.0 81.3	P. ct. 4.8 3.5 1.7 1.9 1.4 1.4 1.4 1.2	Gms. 1,657	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	27	28	28	28	
						P. ct. 10.5 14.1 15.4 16.2 17.1 16.5 16.7 16.0	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 72.7 78.4 76.4 80.6 80.8 80.9 81.0 81.3	P. ct. 4.8 3.5 1.7 1.9 1.4 1.4 1.4 1.2	Gms. 1,657	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	27	28	28	28	
						P. ct. 10.5 14.1 15.4 16.2 17.1 16.5 16.7 16.0	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 72.7 78.4 76.4 80.6 80.8 80.9 81.0 81.3	P. ct. 4.8 3.5 1.7 1.9 1.4 1.4 1.4 1.2	Gms. 1,657	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	27	28	28	28	
						P. ct. 10.5 14.1 15.4 16.2 17.1 16.5 16.7 16.0	P. ct. 3.7 3.4 3.4 3.4 3.4 3.4 3.4 3.4	P. ct. 9.2 8.4 8.4 8.4 8.4 8.4 8.4 8.4	P. ct. 3.2 4 4 4 4 4 4 4	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.2 1.1 1.1 1.1 1.1 1.1 1.1 1.1	P. ct. 72.7 78.4 76.4 80.6 80.8 80.9 81.0 81.3	P. ct. 4.8 3.5 1.7 1.9 1.4 1.4 1.4 1.2	Gms. 1,657	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	27	28	28	28	
182	48.0	11 a. m. to 2.40 p. m.	1.20 to 2.40 p. m.	1.53 to 1.59 p. m.	A B C D E F G H	P. ct. 11.8 12.8 13.4 12.7 12.7 13.3 12.7 12.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 5.9 6.1 5.9 6.3 6.3 6.0 6.7 6.5	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.0 0.8 0.8 0.8 0.8 0.8 0.8 0.8	P. ct. 77.2 80.8 81.0 81.0 81.3 81.3 81.0 81.0	P. ct. 6.9 3.3 1.1 1.0 0.0 0.0 0.0 0.0	Gms. 1,487	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	41	28	28	28	
						P. ct. 11.8 12.8 13.4 12.7 12.7 13.3 12.7 12.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 5.9 6.1 5.9 6.3 6.3 6.0 6.7 6.5	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.0 0.8 0.8 0.8 0.8 0.8 0.8 0.8	P. ct. 77.2 80.8 81.0 81.0 81.3 81.3 81.0 81.0	P. ct. 6.9 3.3 1.1 1.0 0.0 0.0 0.0 0.0	Gms. 1,487	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	41	28	28	28	
						P. ct. 11.8 12.8 13.4 12.7 12.7 13.3 12.7 12.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 5.9 6.1 5.9 6.3 6.3 6.0 6.7 6.5	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.0 0.8 0.8 0.8 0.8 0.8 0.8 0.8	P. ct. 77.2 80.8 81.0 81.0 81.3 81.3 81.0 81.0	P. ct. 6.9 3.3 1.1 1.0 0.0 0.0 0.0 0.0	Gms. 1,487	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	41	28	28	28	
						P. ct. 11.8 12.8 13.4 12.7 12.7 13.3 12.7 12.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 4.1 6.1 5.5 6.3 6.4 6.0 6.7 6.5	P. ct. 5.9 6.1 5.9 6.3 6.3 6.0 6.7 6.5	P. ct. 0.0 0 0 0 0 0 0 0	P. ct. 1.0 0.8 0.8 0.8 0.8 0.8 0.8 0.8	P. ct. 77.2 80.8 81.0 81.0 81.3 81.3 81.0 81.0	P. ct. 6.9 3.3 1.1 1.0 0.0 0.0 0.0 0.0	Gms. 1,487	Lbs. 77.4	P. ct. 32	0.11	0.35	0.40	41	28	28	28	

137	48.0	10.30 a. m. to 2.40 p. m.	1.40 to 2.40 p. m.	2.09 to 2.16....	A 12.4 H 13.5 C 12.7 D 12.8 E 12.8 F 13.2 G 11.5 H	5.0 4.1 6.2 5.8 5.8 5.3 7.3	4.2 0 0 0 0 0 0	1 1 0 0 0 0 0	1 1 0 0 0 0 0	75.2 81.0 81.4 81.3 81.4 81.4 81.5 81.2	4.4 0 0 0 0 0 0	1,003	37	20	30
136	50.2	10.30 a. m. to 4 p. m.	2.40 to 4 p. m.	3.21 to 3.28 p. m.	A 11.1 B 12.4 C 12.1 D 12.1 E 11.7 F 12.4 G 10.7 H 11.1	4.2 6.9 6.4 6.9 7.5 6.9 8.2 8.0	7.4 1.4 7.7 0 0 0 1 0	0 1 1 0 0 0 0 0	0 1 2 1 1 1 1 0	2.1 75.2 80.2 80.8 80.7 80.6 81.0 80.9	9.5 1.9 1.0 1.2 1.1 1.1 1.1 0	1,582	33	30	43
140	53.8	10.30 a. m. to 3.20 p. m.	2 to 3.20 p. m.	2.31 to 2.38 p. m.	A 11.2 B 11.7 C 11.3 D 10.2 E 9.2 F 9.2 G 11.7 H 14.1 14.0 15.1 15.6 15.5	7.0 7.8 8.2 9.3 10.4 10.4 2.8 3.2 3.0 2.6 1.9 2.0	2.2 1 0 0 0 0 7.6 2.1 1.6 1.0 1.1	0 0 0 0 0 0 1 0 0 0 0 0	0 0 0 0 0 0 1.8 1.0 1.7 3 5 3	.5 80.4 80.4 80.5 80.4 80.9 81.1 81.1	2.7 1 0 0 0 0 9.5 3.1 2.6 1.4 1.5 1.4	1,545	39	37	54
161	58.2	11.20 a. m. to 4 p. m.	2.20 to 4 p. m.	2.34 to 3.39 p. m.	A 11.7 B 14.1 C 14.0 D 15.1 E 15.6 F 15.5 G 15.9 H	2.8 3.2 3.0 2.6 1.9 2.0	7.6 2.1 1.6 1.0 1.1	0 0 0 0 0 0	0 1.8 1.0 1.7 3 5 3	80.4 76.0 79.6 80.4 80.9 81.1 81.1	0 9.5 3.1 2.6 1.4 1.5 1.4	1,228	55	19	30
150	58.9	11 a. m. to 2.40 p. m.	1.40 to 2.40 p. m.	2.07 to 2.14 p. m.	A 10.3 B 12.7 C 13.4 D 13.6 E 13.3 F 13.8 G 13.9 H 14.4	4.6 4.9 4.5 4.6 4.3 4.0 3.9 4.1	8.0 1.9 1.3 9 1.2 1.0 8 2	0 0 0 0 0 0 0 0	0 1.9 5 2 2 1 1 0	1.9 75.2 78.9 80.6 80.7 81.1 81.1 81.3	9.9 2.5 1.5 1.1 1.1 1.1 9	1,546	49	32	37
138	57.1	10.45 a. m. to 3 p. m.	1.20 to 3 p. m.	2.08 to 2.15 p. m.	A 12.1 B 14.8 C 14.9 D 14.4 E 14.4 F 14.4 G 13.4 H	3.7 3.4 3.7 4.4 4.2 4.2	6.3 9 2 0 0 0	1 1 1 0 0 0	1 1.2 0 0 0 0	1.2 76.7 80.7 81.1 81.3 81.3	7.5 1.1 1.1 1.1 1.1 1	1,634	55	26	37
					A 13.4 B 13.4 C 13.4 D 13.4 E 13.4 F 13.4 G 13.4 H	5.5 5.5 5.5 5.5 5.5 5.5	0 0 0 0 0 0	0 0 0 0 0 0	0 0 0 0 0 0	0 81.1 81.1 81.1 81.1 81.1	0 0 0 0 0 0	1,335	15.8	35.5	

• Doubtful.

TABLE 3.—Results of combustion tests—Continued.
ILLINOIS COAL.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperatures and pressures.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.										Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.	Temperature.	Weight of air per pound of combustible.	Excess of air.	Gas pressure, inches of water.			
						CO ₂ .	Unsaturated hydrocarbons.	O ₂ .	CO.	CH ₄ .	H ₂ .	N ₂ .	Total gaseous combustible.	15	16						17	18	19	20
1	2	3	4	5	6	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	Gms.	° C.	Lbs.	P. ct.	In ash pit, above atmospheric.	Over fuel bed, below atmospheric.	At rear of chamber, below atmospheric.	In uptake, below atmospheric.		
166	23.0	11.40 a. m. to 3.40 p. m.	2.20 to 3.40 p. m.	2.36 to 3.21 p. m.	A B C D E F G H	7.7	8.9	6.5	0.1	2.3	74.5	11.3	2.7	1.6	1.8	1,332	14.7	37.3	0.0	0.12	0.14			
						11.2	6.3	2.0	0.7	70.8	2.7	1.6	1.8	1.1	0.6	80.5	1.1	0.6	80.5	0.0	0.13	0.15		
						12.3	5.0	1.2	0.4	80.7	1.6	0.0	0.0	0.0	0.0	80.7	1.1	0.0	80.7	0.0	0.12	0.14		
						12.7	6.1	1.4	0.0	81.1	0.0	0.0	0.0	0.0	0.0	80.8	1.1	0.0	80.8	0.0	0.12	0.14		
168	20.0	11.40 a. m. to 3.40 p. m.	2 to 3.40 p. m.	2.24 to 3.09 p. m.	A B C D E F G H	9.2	1.5	9.6	0.0	3	70.4	1.8	0.0	1.8	1,332	14.7	37.3	0.0	0.13	0.15				
						10.4	0.9	8.4	0.0	2	80.1	1.1	0.0	0.0	0.0	80.1	1.1	0.0	80.1	0.0	0.13	0.15		
						10.6	0.0	8.3	0.0	2	80.5	1.1	0.0	0.0	0.0	80.5	1.1	0.0	80.5	0.0	0.13	0.15		
						10.9	0.0	8.5	0.0	2	80.7	1.1	0.0	0.0	0.0	80.7	1.1	0.0	80.7	0.0	0.13	0.15		
167	19.6	11.40 a. m. to 3.20 p. m.	1.40 to 3.20 p. m.	2.06 to 2.53 p. m.	A B C D E F G H	11.2	7.9	7.9	1.1	0.0	80.8	1.1	0.0	939	17.3	61.6	0.0	0.11	0.13					
						10.8	0.0	8.4	0.0	0.0	80.8	1.1	0.0	0.0	0.0	80.8	1.1	0.0	80.8	0.0	0.11	0.13		
						11.2	7.9	7.9	1.1	0.0	80.8	1.1	0.0	0.0	0.0	80.8	1.1	0.0	80.8	0.0	0.11	0.13		
						8.3	8.6	8.6	6.1	0.0	76.4	6.7	0.0	0.0	0.0	76.4	6.7	0.0	76.4	0.0	0.11	0.13		

TABLE 3.—*Results of combustion tests—Continued.*

ILLINOIS COAL—Continued.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperature and pressure.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.										Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.	Temperature.	Weight of air per pound of combustible.	Excess of air.	Gas pressure, inches of water.				
						CO ₂	Unsaturated hydrocarbons.	O ₂	CO.	CH ₄	H ₂	N ₂	Total gaseous combustible.	In ash pit, above atmospheric.	Over fuel bed, below atmospheric.						At rear of chamber, below atmospheric.	In uptake, below atmospheric.			
173	38.8	11.40 a. m. to 4 p. m.	2.40 to 4 p. m.	2.55 to 3.40 p. m.	A B C D C	P. cl. 4.8	P. cl. 10.4	P. cl. 8.5	P. cl. 0.0	P. cl. 1.8	P. cl. 74.5	P. cl. 10.3	P. cl. 1.3	Gms. 1,438						0.12	0.36	0.39			
						9.7	9.0	1.1	0.0	0.0	0.0	80.6	1.3												
						10.1	9.0	0.3	0.0	0.0	0.0	80.5	0.0												
						9.8	9.6	1.1	0.0	0.0	0.0	80.5	0.0												
						9.8	9.6	1.1	0.0	0.0	0.0	80.5	0.0												
175	50.0	11.40 a. m. to 3.40 p. m.	2 to 3.40 p. m.	2.30 to 3.15 p. m.	A B C D C	P. cl. 10.5	P. cl. 13.2	P. cl. 5.1	P. cl. 4.9	P. cl. 1.0	P. cl. 77.5	P. cl. 5.9	P. cl. 1.3	Gms. 1,516						.35	.25	.31	0.60		
						12.7	12.7	4.8	0.0	0.0	0.0	80.6	0.0												
						14.1	14.1	4.4	0.0	0.0	0.0	80.8	0.0												
						14.0	14.0	4.5	0.0	0.0	0.0	80.8	0.0												
						13.8	13.8	5.0	0.0	0.0	0.0	80.8	0.0												
174	50.1	11.40 a. m. to 3.20 p. m.	2 to 3.20 p. m.	2.19 to 3.04 p. m.	A B C D C	P. cl. 14.0	P. cl. 14.1	P. cl. 4.7	P. cl. 4.2	P. cl. 0.0	P. cl. 81.0	P. cl. 2.2	P. cl. 1.1	Gms. 1,216											
						14.1	14.1	4.7	0.0	0.0	0.0	81.1	0.0												
						14.0	14.0	4.8	0.0	0.0	0.0	81.1	0.0												
						13.8	13.8	4.8	0.0	0.0	0.0	80.9	0.0												
						12.7	12.7	4.8	0.0	0.0	0.0	80.9	0.0												
					H G A E D C	P. cl. 7.7	P. cl. 11.2	P. cl. 6.9	P. cl. 5.8	P. cl. 1.0	P. cl. 76.5	P. cl. 6.9	P. cl. 2.1	Gms. 1,522						.30	.25	.32	.77		
						11.2	11.2	6.9	1.8	0.0	0.0	76.8	2.1												
						11.6	11.6	6.6	1.1	0.0	0.0	80.4	1.4												
						12.3	12.3	6.3	0.9	0.0	0.0	80.3	1.2												
						12.8	12.8	6.7	0.9	0.0	0.0	80.3	1.1												
					H G A E D C	P. cl. 12.8	P. cl. 12.8	P. cl. 6.8	P. cl. 5.8	P. cl. 0.0	P. cl. 80.9	P. cl. 0.5	P. cl. 0.7	Gms. 1,245											
						12.7	12.7	6.8	0.9	0.0	0.0	80.9	0.5												
						12.7	12.7	6.8	0.9	0.0	0.0	80.9	0.5												
						12.7	12.7	6.8	0.9	0.0	0.0	80.9	0.5												
						12.8	12.8	6.1	0.9	0.0	0.0	80.9	0.7												
					H G A E D C	P. cl. 12.8	P. cl. 12.8	P. cl. 6.0	P. cl. 5.8	P. cl. 0.0	P. cl. 80.9	P. cl. 0.3	P. cl. 0.3	Gms. 1,084											
						10.1	10.1	9.1	0.0	0.0	0.0	80.4	0.0												
						10.0	10.0	9.3	0.0	0.0	0.0	80.9	0.0												
						9.8	9.8	9.7	0.0	0.0	0.0	80.5	0.0												
						9.8	9.8	9.6	1.1	0.0	0.0	80.5	0.0												

[illegible]

TABLE 3.—Results of combustion tests—Continued.

PITTSBURGH SCREENINGS.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperatures and pressures.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.										Weight of ash per pound of combustible.	Excess of air.	Gas pressure, inches of water.			
						CO ₂	Unsaturated hydrocarbons.	O ₂	CO.	CH ₄	H ₂	N ₂	Total gaseous combustible.	Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.			In ash pit, above atmospheric.	Over fuel bed, below atmospheric.	At rear of chamber, below atmospheric.	In uptake, below atmospheric.
195	24.0	11 a. m. to 4 p. m.	2.40 to 4 p. m.	3.10 to 3.32 p. m.	B	P. ct. 8.9 P. 10.8	P. ct. 0.1 P. 0.8	P. ct. 1.2 P. 2.5	P. ct. 14.6 P. 28.6	P. ct. 0.6 P. 1.4	P. ct. 5.7 P. 11.4	P. ct. 75.6 P. 78.3	P. ct. 11.2 P. 11.2	Gms. 0.054 Gms. 0.180	1.476	1.255	P. ct. 5.3	0.07	0.11	0.14	0.20
						P. ct. 13.5 P. 15.2	P. ct. 2.8 P. 3.1	P. ct. 2.0 P. 2.0	P. ct. 4.6 P. 9.9	P. ct. 2.2 P. 3.0	P. ct. 2.2 P. 3.0	P. ct. 78.3 P. 81.0	P. ct. 6.2 P. 11.2	Gms. 0.004 Gms. 0.128	1.476	1.255	P. ct. 5.3	0.07	0.11	0.14	0.20
						P. ct. 15.8 P. 15.9	P. ct. 2.8 P. 3.1	P. ct. 2.0 P. 2.0	P. ct. 4.6 P. 9.9	P. ct. 2.2 P. 3.0	P. ct. 2.2 P. 3.0	P. ct. 78.3 P. 81.0	P. ct. 6.2 P. 11.2	Gms. 0.004 Gms. 0.128	1.476	1.255	P. ct. 5.3	0.07	0.11	0.14	0.20
						P. ct. 15.9 P. 15.9	P. ct. 2.8 P. 3.1	P. ct. 2.0 P. 2.0	P. ct. 4.6 P. 9.9	P. ct. 2.2 P. 3.0	P. ct. 2.2 P. 3.0	P. ct. 78.3 P. 81.0	P. ct. 6.2 P. 11.2	Gms. 0.004 Gms. 0.128	1.476	1.255	P. ct. 5.3	0.07	0.11	0.14	0.20
207	21.4	11 a. m. to 3.40 p. m.	2.30 to 3.40 p. m.	2.47 to 3.07 p. m.	B	P. ct. 9.0 P. 10.2	P. ct. 0.4 P. 0.6	P. ct. 4.4 P. 5.2	P. ct. 8.1 P. 12.6	P. ct. 2.8 P. 3.4	P. ct. 2.7 P. 3.4	P. ct. 75.0 P. 78.7	P. ct. 11.6 P. 11.6	Gms. 0.073 Gms. 0.089	1.331	1.270	P. ct. 5.3	0.01	0.15	0.19	0.30
						P. ct. 11.9 P. 11.9	P. ct. 6.8 P. 7.4	P. ct. 6.8 P. 7.4	P. ct. 2.4 P. 11.4	P. ct. 2.4 P. 11.4	P. ct. 2.4 P. 11.4	P. ct. 81.2 P. 81.1	P. ct. 3.0 P. 11.1	Gms. 0.001 Gms. 0.064	1.331	1.270	P. ct. 5.3	0.01	0.15	0.19	0.30
						P. ct. 11.4 P. 11.4	P. ct. 7.4 P. 7.4	P. ct. 7.4 P. 7.4	P. ct. 11.4 P. 11.4	P. ct. 11.4 P. 11.4	P. ct. 11.4 P. 11.4	P. ct. 81.1 P. 81.1	P. ct. 11.1 P. 11.1	Gms. 0.064 Gms. 0.064	1.331	1.270	P. ct. 5.3	0.01	0.15	0.19	0.30
						P. ct. 11.2 P. 11.2	P. ct. 7.8 P. 7.8	P. ct. 7.8 P. 7.8	P. ct. 11.2 P. 11.2	P. ct. 11.2 P. 11.2	P. ct. 11.2 P. 11.2	P. ct. 80.9 P. 80.9	P. ct. 11.1 P. 11.1	Gms. 0.020 Gms. 0.094	1.331	1.270	P. ct. 5.3	0.01	0.15	0.19	0.30
208	21.4	11.20 a. m. to 4 p. m.	3 to 4 p. m.	3.26 to 3.36 p. m.	B	P. ct. 10.0 P. 10.0	P. ct. 1.1 P. 1.1	P. ct. 4.7 P. 5.0	P. ct. 8.0 P. 12.6	P. ct. 4.4 P. 12.6	P. ct. 4.4 P. 12.6	P. ct. 75.3 P. 78.1	P. ct. 10.0 P. 11.6	Gms. 0.094 Gms. 0.075	1.346	1.246	P. ct. 5.3	0.03	0.14	0.17	0.37
						P. ct. 12.5 P. 12.5	P. ct. 2.8 P. 2.8	P. ct. 5.0 P. 5.0	P. ct. 2.8 P. 12.6	P. ct. 2.8 P. 12.6	P. ct. 2.8 P. 12.6	P. ct. 81.0 P. 81.0	P. ct. 3.2 P. 11.1	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.03	0.14	0.17	0.37
						P. ct. 10.3 P. 10.3	P. ct. 8.6 P. 8.6	P. ct. 8.6 P. 8.6	P. ct. 2.8 P. 12.6	P. ct. 2.8 P. 12.6	P. ct. 2.8 P. 12.6	P. ct. 81.0 P. 81.0	P. ct. 3.2 P. 11.1	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.03	0.14	0.17	0.37
						P. ct. 9.5 P. 9.5	P. ct. 9.8 P. 9.8	P. ct. 9.8 P. 9.8	P. ct. 2.8 P. 12.6	P. ct. 2.8 P. 12.6	P. ct. 2.8 P. 12.6	P. ct. 80.7 P. 80.7	P. ct. 0.0 P. 11.1	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.03	0.14	0.17	0.37
195	22.1	10.40 a. m. to 4.43 p. m.	3 to 4.40 p. m.	3.36 to 3.59 p. m.	B	P. ct. 10.2 P. 10.2	P. ct. 8.0 P. 8.4	P. ct. 8.0 P. 8.4	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 78.6 P. 81.2	P. ct. 8.3 P. 11.1	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.14	0.09	0.18	0.68
						P. ct. 10.1 P. 10.1	P. ct. 9.3 P. 9.3	P. ct. 9.3 P. 9.3	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 80.4 P. 80.4	P. ct. 11.2 P. 11.2	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.14	0.09	0.18	0.68
						P. ct. 7.9 P. 7.9	P. ct. 11.8 P. 11.8	P. ct. 11.8 P. 11.8	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 80.5 P. 80.5	P. ct. 0.0 P. 11.1	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.14	0.09	0.18	0.68
						P. ct. 8.8 P. 8.8	P. ct. 10.7 P. 10.7	P. ct. 10.7 P. 10.7	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 2.7 P. 12.6	P. ct. 80.5 P. 80.5	P. ct. 0.0 P. 11.1	Gms. 0.059 Gms. 0.059	1.346	1.246	P. ct. 5.3	0.14	0.09	0.18	0.68

TABLE 3.—*Results of combustion tests—Continued.*

PITTSBURGH SCREENINGS—Continued.

Test No.	Coal fired per square foot of grate area per hour.	Duration of test.	Period used for obtaining averages of gas temperature and pressures.	Period of gas sampling.	Section of furnace where samples were taken.	Average composition of gases, by volume.										Weight of tar per cubic foot of gas.	Weight of soot per cubic foot of gas.	Temperature.	Weight of air per pound of combustible.	Excess of air.	Gas pressure, inches of water.					
						CO ₂	Unsaturated hydrocarbons.	O ₂	CO.	CH ₄	H ₂	N ₂	Total gaseous combustible.	15	16						17	18	19	20	21	22
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23				
196	39.2	12 m. to 3.40 p. m.	2.40 to 3.40 p. m.	3 to 3.21 p. m.	0 11 A C E G	P. ct.	10.4	3.6	8.4	0.4	2.5	74.7	11.2	0.024	0.113	1,610	14.1	25.7	30.1	0.30	0.13	0.25	1.26			
						P. ct.	10.1	6.0	4.1	0.2	1.1	78.5	5.4	0.014	0.160	1,427	3.9	0.014	0.160	1,350	14.1	25.7	30.1	0.25	1.26	
						P. ct.	13.2	3.9	2.8	0.0	1.1	79.0	3.7	0.014	0.160	1,427	3.9	0.014	0.160	1,350	14.1	25.7	30.1	0.13	0.25	1.26
						P. ct.	13.8	4.3	2.8	0.0	1.1	81.2	4.4	0.014	0.160	1,427	3.7	0.014	0.160	1,350	14.1	25.7	30.1	0.13	0.25	1.26
						P. ct.	14.1	4.2	4.0	0.0	0.0	81.5	4.4	0.014	0.160	1,427	4.0	0.014	0.160	1,350	14.1	25.7	30.1	0.13	0.25	1.26
197	35.6	11 a. m. to 4 p. m.	2 to 4 p. m.	2.52 to 3.12 p. m.	0 11 A C E G	P. ct.	7.3	0.9	1.3	16.4	8.7	65.8	25.7	2.253	272	1,530	14.5	28.0	30.1	0.30	0.19	0.27	1.05			
						P. ct.	11.6	2.9	6.4	4.4	3.3	75.4	10.1	0.006	0.096	1,530	4.6	0.006	0.096	1,280	14.5	28.0	30.1	0.19	0.27	1.05
						P. ct.	13.5	3.2	3.1	0.0	1.5	76.7	4.6	0.006	0.096	1,530	4.6	0.006	0.096	1,280	14.5	28.0	30.1	0.19	0.27	1.05
						P. ct.	13.6	4.9	4.0	0.0	0.0	81.1	4.4	0.006	0.096	1,530	4.4	0.006	0.096	1,280	14.5	28.0	30.1	0.19	0.27	1.05
						P. ct.	13.8	4.8	4.0	0.0	0.0	81.4	0.0	0.006	0.096	1,530	4.4	0.006	0.096	1,280	14.5	28.0	30.1	0.19	0.27	1.05
198	42.0	11 a. m. to 3.40 p. m.	2.40 to 3.40 p. m.	2.57 to 3.17 p. m.	0 11 A C E G	P. ct.	6.7	5.7	12.7	1.1	2.2	72.6	15.0	0.025	0.099	1,243	14.5	28.0	30.1	0.48	0.24	0.24	1.04			
						P. ct.	10.1	4.0	7.8	0.0	1.6	76.4	9.5	0.006	0.143	1,243	3.9	0.006	0.143	1,243	14.5	28.0	30.1	0.24	0.24	1.04
						P. ct.	13.2	3.4	2.6	0.0	1.2	76.5	3.4	0.002	0.102	1,566	4.4	0.002	0.102	1,243	14.5	28.0	30.1	0.24	0.24	1.04
						P. ct.	13.3	4.8	3.0	0.0	1.1	81.5	3.0	0.002	0.102	1,566	4.4	0.002	0.102	1,243	14.5	28.0	30.1	0.24	0.24	1.04
						P. ct.	13.8	4.4	3.0	0.0	0.0	81.6	2.2	0.002	0.102	1,566	4.4	0.002	0.102	1,243	14.5	28.0	30.1	0.24	0.24	1.04
205	35.2	11.20 a. m. to 4.20 p. m.	3.07 to 4.20 p. m.	3.23 to 3.55 p. m.	0 11 A C E G	P. ct.	8.6	3.8	15.3	2.2	3.4	70.4	19.2	0.088	0.171	1,495	14.5	28.5	30.1	0.21	0.25	0.25	0.73			
						P. ct.	10.3	2.1	10.0	3.1	3.2	74.1	13.5	0.027	0.272	1,495	1.8	0.027	0.272	1,495	14.5	28.5	30.1	0.25	0.25	0.73
						P. ct.	12.8	1.7	7.5	0.0	1.9	77.0	8.5	0.008	0.080	1,300	1.0	0.008	0.080	1,300	14.5	28.5	30.1	0.25	0.25	0.73
						P. ct.	12.1	6.2	8.0	0.0	2.2	80.7	1.8	0.008	0.080	1,300	1.0	0.008	0.080	1,300	14.5	28.5	30.1	0.25	0.25	0.73
						P. ct.	12.8	4.8	3.0	0.0	1.1	81.1	4.8	0.008	0.080	1,294	1.0	0.008	0.080	1,294	14.5	28.5	30.1	0.25	0.25	0.73
206	35.2	11.20 a. m. to 4.20 p. m.	3.07 to 4.20 p. m.	3.23 to 3.55 p. m.	0 11 A C E G	P. ct.	12.4	6.5	1.1	0.0	0.0	81.0	1.1	1.1	1,145	16.2	43.8	43.8	0.1	0.1	0.1	0.1				
						P. ct.	12.4	6.5	1.1	0.0	0.0	81.0	1.1	1.1	1,145	16.2	43.8	43.8	0.1	0.1	0.1	0.1				
						P. ct.	12.4	6.5	1.1	0.0	0.0	81.0	1.1	1.1	1,145	16.2	43.8	43.8	0.1	0.1	0.1	0.1				
						P. ct.	12.4	6.5	1.1	0.0	0.0	81.0	1.1	1.1	1,145	16.2	43.8	43.8	0.1	0.1	0.1	0.1				
						P. ct.	12.4	6.5	1.1	0.0	0.0	81.0	1.1	1.1	1,145	16.2	43.8	43.8	0.1	0.1	0.1	0.1				

194	44.2	11.20 a. m. to 3.40 p. m.	2.40 to 3.40 p. m.	3.01 to 3.21 p. m.	0 11 C G	0 11 C G	0.3 11.6 9.7 10.6	4.3 7.0 9.6 8.7	9.6 7.0 9.6 8.7	1 0 0 0	2.1 74.7 80.7 80.7	11.7 2.4 0 0	0.032 0.071 0.016 0	25 40 50 1.28
204	37.9	11.20 a. m. to 3.40 p. m.	2.40 to 3.40 p. m.	2.55 to 4.04 p. m.	0 11 A C E G	0 11 A C E G	0.8 9.8 12.5 10.1 9.8	5.3 7.8 6.1 8.5 9.1	9.3 2.5 2.6 0 0	2 1 0 0 0	1.6 74.9 78.9 81.0 80.8	11.3 8.5 3.0 0 0	0.073 0.145 0.033 1.244 1.132 1.070	21 18 25 78 83.0 83.0
199	47.1	11 a. m. to 3.40 p. m.	2 to 3.40 p. m.	2.39 to 3 p. m.	0 11 A C E G	0 11 A C E G	0.6 11.1 12.5 15.0 15.8	3.4 4.6 1.8 2.3 2.4	13.4 3.3 6.6 1.6 3	1 0 1 0 0	9.0 78.9 77.4 80.9 81.4	23.5 8.4 0.3 1.8 4	0.056 0.173 0.033 0.081 1.422	40 20 30 74 12.2 12.2
192	45.3	11 a. m. to 3.20 p. m.	2.20 to 3.20 p. m.	2.42 to 3.02 p. m.	0 11 A C E G	0 11 A C E G	0.4 10.4 11.6 13.7 14.1 13.6	2.8 5.3 3.9 4.1 4.2 4.7	17.4 4.8 4.4 8 2 1	2 1 1 0 0 0	3.4 69.8 78.2 78.9 81.4 81.6	21.0 6.1 0.3 5.6 1.1 1	0.031 0.057 0.111 0.091 1.331 1.283 1.271	44 36 43 90 32.2 32.2
200	50.5	11 a. m. to 3.40 p. m.	2.40 to 4 p. m.	2.58 to 3.18 p. m.	0 11 A C E G H	0 11 A C E G H	0.8 9.8 14.5 11.5 11.4 10.9	3.1 7.7 4.6 7.4 7.5 8.1	4.2 6 3.5 1 0 1	1.2 0 1 0 0 0	3.6 74.8 81.4 76.3 81.0 81.1 80.9	9.3 1.1 0.3 4.6 0 0 1	0.079 0.032 0.035 0.041 0 0 1.120 1.106	52 40 54 1.06 83.2 83.2
201	60.0	11.20 a. m. to 4.20 p. m.	3 to 4.20 p. m.	3.36 to 3.56 p. m.	0 11 A C E G H	0 11 A C E G H	0.3 12.3 13.4 13.5 13.9 13.4	3.8 3.7 2.5 5.0 4.7 5.3	8.8 3.9 3.1 2 2 0	4 1 1 0 0 0	3.1 74.5 79.6 78.9 81.2 81.3	12.4 4.4 0.3 6.2 2 0	0.032 0.112 0.033 0.034 0.032 0.034 1.274	45 28 34 76 84.6 84.6
202	57.3	11.20 a. m. to 4.30 p. m.	3.40 to 4.30 p. m.	3.57 to 4.18 p. m.	0 11 A C E G H	0 11 A C E G H	0.8 10.1 12.8 12.8 12.4 12.6	6.1 4.9 2.1 0 6.4 6.3	9.1 9.1 7.1 0 0 0	4 2 1 0 0 0	3.2 73.3 76.7 78.9 81.3 81.1	12.8 9.2 0.3 2.9 0 0	0.038 0.066 0.033 0.045 0.032 0.045 1.180	50 33 35 69 42.8 42.8

DISCUSSION OF COMBUSTION PROCESS IN COMBUSTION SPACE.

The combustion process in the combustion space above and beyond the fuel bed is well illustrated in figure 13, which gives the percentage of the main constituents of the furnace gases at various sections of the long combustion chamber. The figure represents the actual conditions of test 187. The ordinates are the percentages of gases by volume; the abscissa has two scales—namely, volume of combustion

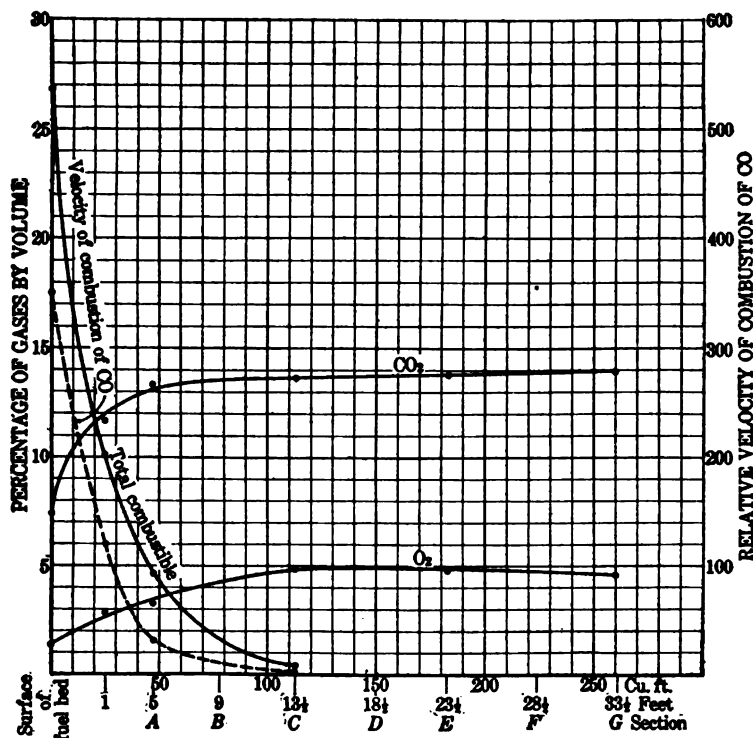


FIGURE 13.—Progress of composition of furnace gases at various distances above the fuel bed. Test 187, Pittsburgh screenings; rate of combustion, 35.6; sampling period, 20 minutes.

space in cubic feet and the average length of gas travel from the surface of the fuel bed. The points representing the composition of gases at the surface of the fuel bed are the averages of seven samples; at 1 foot from the surface of fuel bed, eight samples; at A, six samples; and at C, E, and G, nine samples. All of these 48 gas samples were taken simultaneously over a period of 20 minutes. The rate of combustion was 35.6 pounds of coal per square foot of grate area. The coal used was Pittsburgh screenings. The curve labeled "total combustible" represents the sum of combustible gases as given in Table 3, column 14.

The curves show that the gases leaving the fuel bed contain over 25 per cent of combustible gases, about one per cent of O_2 , and 7 per cent of CO_2 . Before these gases reached section A, enough air was added to make the total air supply exceed the amount theoretically needed by 19 per cent. Most of this air was added through the tuyères very near to the surface of the fuel bed, and in a way that facilitated its mixing with the combustible gases rising from the fuel bed. The mixing is illustrated in figure 7. The progress of combustion is indicated by the drop of the total combustible curve and the rise of the CO_2 curve. The change in the composition of the gases indicated by these two curves show that the combustion is most rapid within a foot from the surface of the fuel bed; as the gases pass farther away from the fuel bed the rapidity of combustion decreases. At a distance of $13\frac{1}{2}$ feet from the grate there is very little combustible gas left, showing that combustion is practically complete.

The length or the volume of the combustion space required for practically complete combustion seems to depend chiefly on the percentage of excess air, the rate of combustion, and the kind of coal. It is mainly these three factors that have been investigated in the series of tests reported in this bulletin.

The effect of these three factors on the length or volume of the combustion space required for practically complete combustion is shown in a general way in figures 14 to 18, inclusive. These figures are compiled from the 100 tests shown in Table 3. The curves were plotted with the percentage of gaseous combustible in furnace gases as ordinates and the length and volume of combustion space as abscissas. In these figures and those that follow the length of the combustion space in feet is shown at the bottom of the figure, as are the corresponding volumes in cubic feet. Near the fuel bed the volume is much larger per foot of length of gas travel than it is in the uniform cross section of the combustion chamber.

The tests are grouped according to the kind of coal, and according to the rate of combustion. Within each group having the same rate of combustion, individual tests were made with different percentages of excess air. To prevent the diagrams from being crowded, only the curves giving the total combustible gases were plotted, those showing CO_2 and O_2 being omitted, and when more than one test was run with nearly the same excess of air, a curve giving the average of these tests has been plotted instead of each test being plotted separately. The general shape of the curves is the same as that of the combustible curve shown in figure 13. The rapidity of combustion is highest near the fuel bed, and decreases as the gases flow farther away.

Comparison of the different curves showing the same rate of combustion indicate that when the air excess is large the proportion of combustible gases is less at any given cross section of the furnace,

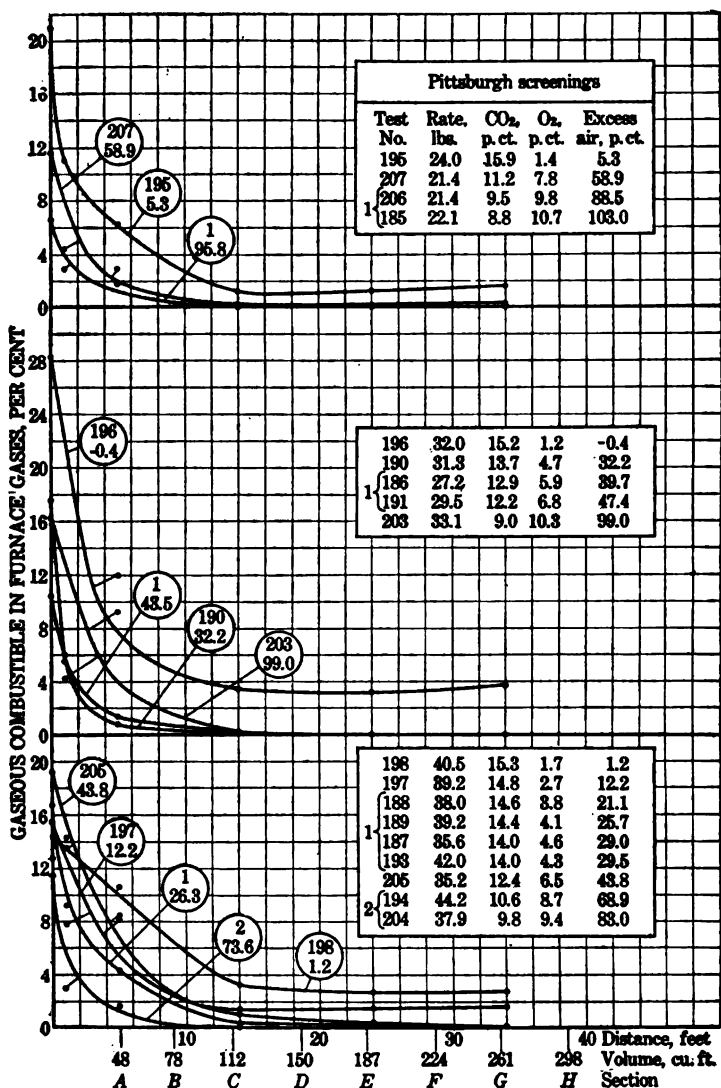


FIGURE 14.—Relation between the percentage of gaseous combustible in furnace gases and the length and volume of combustion space, as shown by tests made with Pittsburgh screenings at rates of combustion of approximately 20, 30, and 40 pounds.

and the combustion is practically complete in a smaller combustion space than when the excess of air is small. Some of the curves and individual points fall out of place. The three main causes of these irregularities are (a) lack of constancy in mixing of the com-

bustible rising from the fuel bed with the air supplied over the fire; (b) the difficulty of obtaining a fair average sample; and (c) the fact that the rate of combustion during the short sampling period may not have been the same as the average rate of the whole test. These three factors are discussed below.

Under normal conditions the air supplied over the fuel bed is introduced through the tuyères as a large number of small jets.

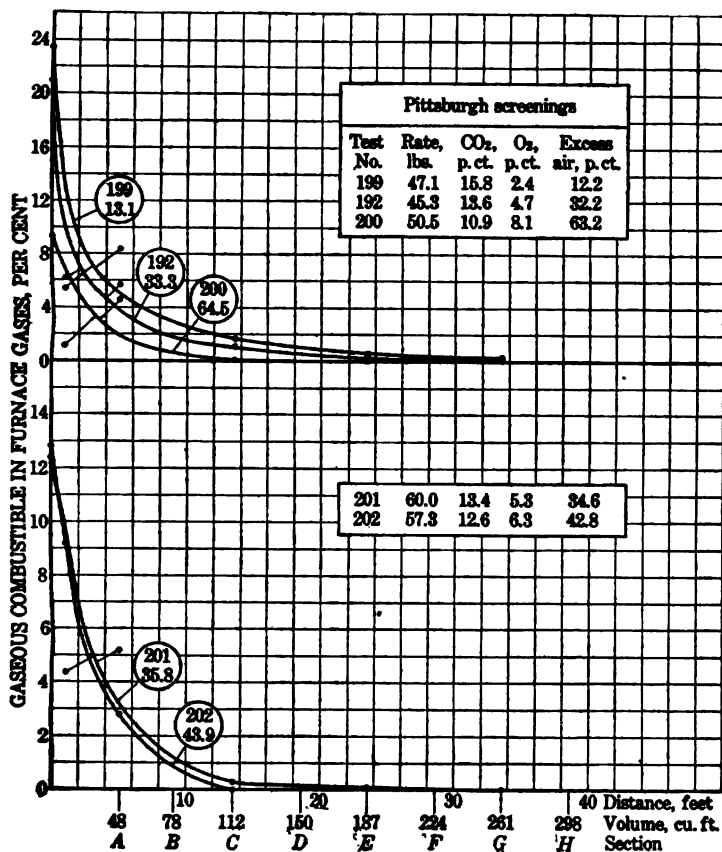


FIGURE 15.—Relation between the percentage of gaseous combustible in furnace gases and the length and volume of combustion space, as shown by tests made with Pittsburgh screenings at rates of combustion of approximately 50 and 60 pounds.

This method of introducing the air produces relatively good mixing. When, however, all the air can not be supplied through the tuyères and part of it must be introduced either through the firing door or through the coal magazines, the air thus admitted enters in large streams and does not mix readily with the combustible; the gases tend to remain stratified and in spite of the large excess of air the combustion does not proceed much faster than when the air is introduced only through the tuyères.

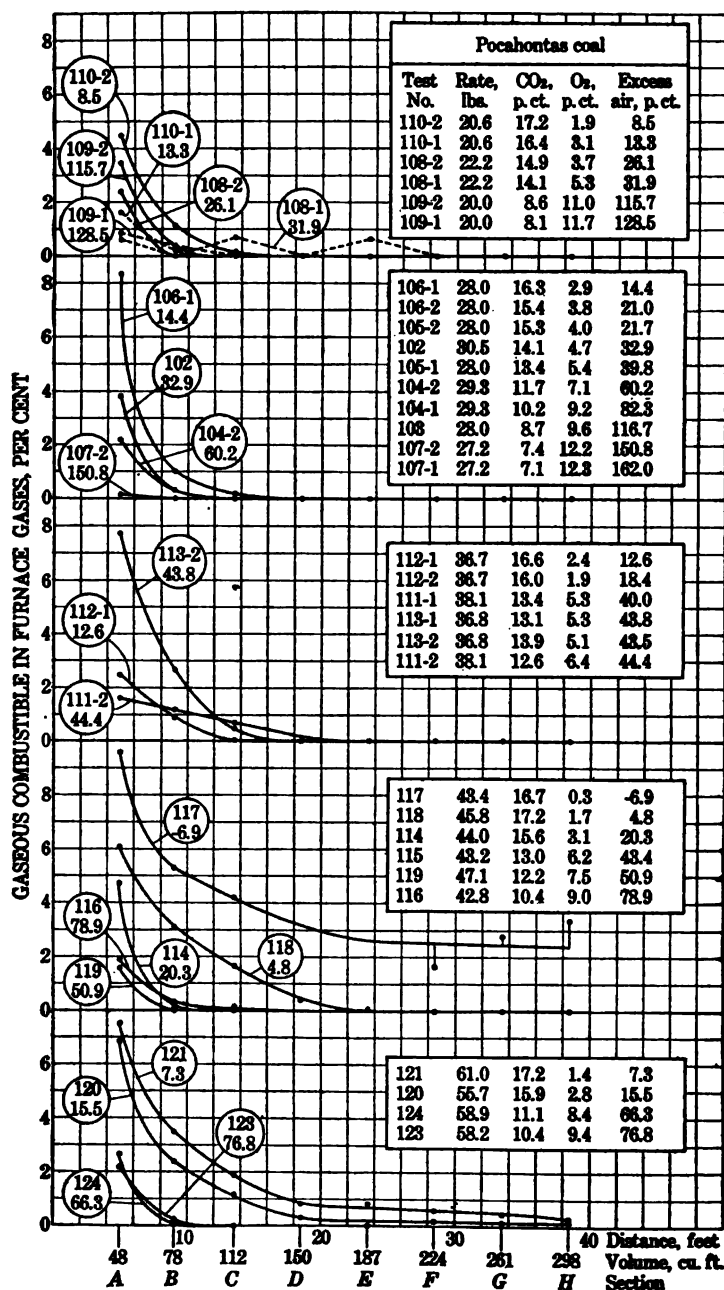


FIGURE 16.—Relation between the percentage of gaseous combustible in furnace gases and the length and volume of combustion space, as shown by tests made with Pocahontas coal at rates of combustion of approximately 20, 30, 40, 50, and 60 pounds.

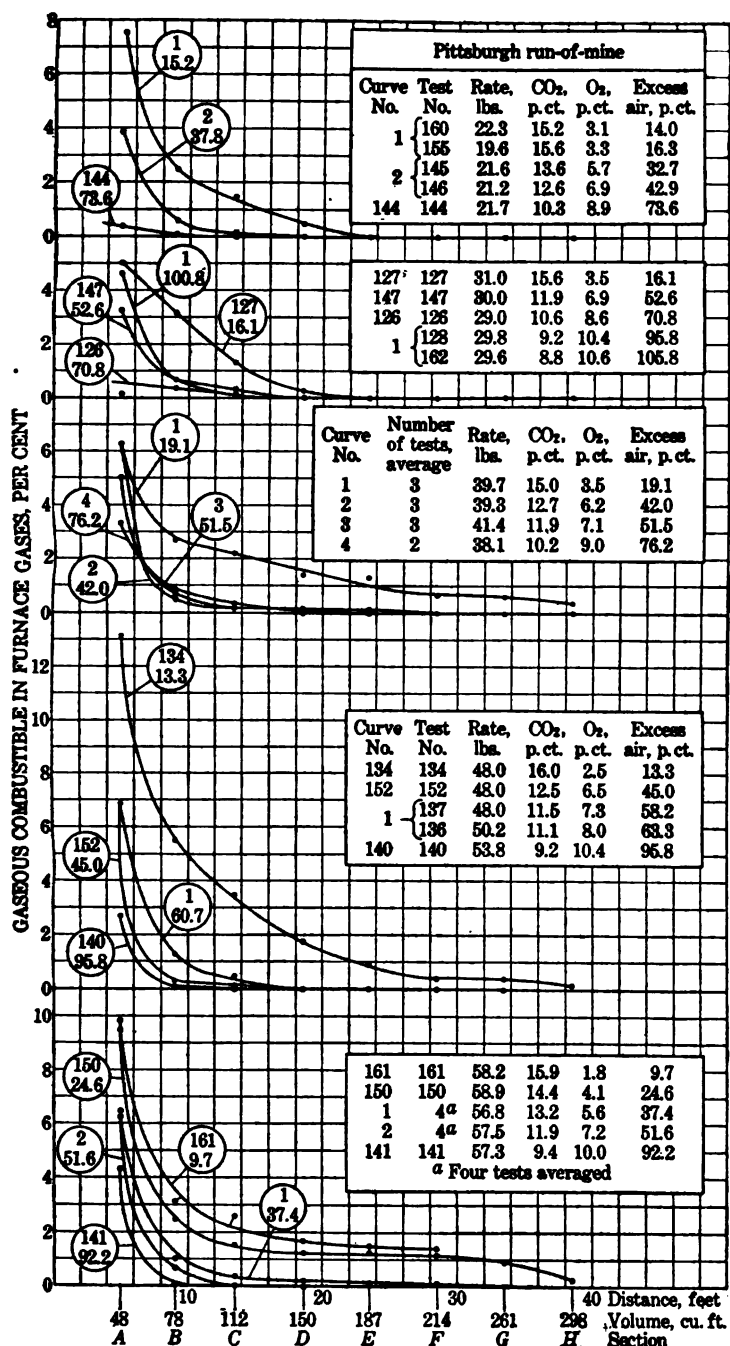


FIGURE 17.—Relation between the percentage of gaseous combustibles in furnace gases and the length and volume of combustion space, as shown by tests made with Pittsburgh run-of-mine coal at rates of combustion of approximately 20, 30, 40, 50 and 60 pounds.

Obtaining a fair average sample of furnace gases from the space over the fuel bed, where the gases are greatly stratified, is particularly difficult. This difficulty can be better appreciated by examining figure 7 (p. 17) which shows the streams of gases and the position of gas samplers over the fuel bed. The upper holes of the two side rows of gas samplers are in the path of the air coming in through the tuyères. As a result of this position the average of the gas samples

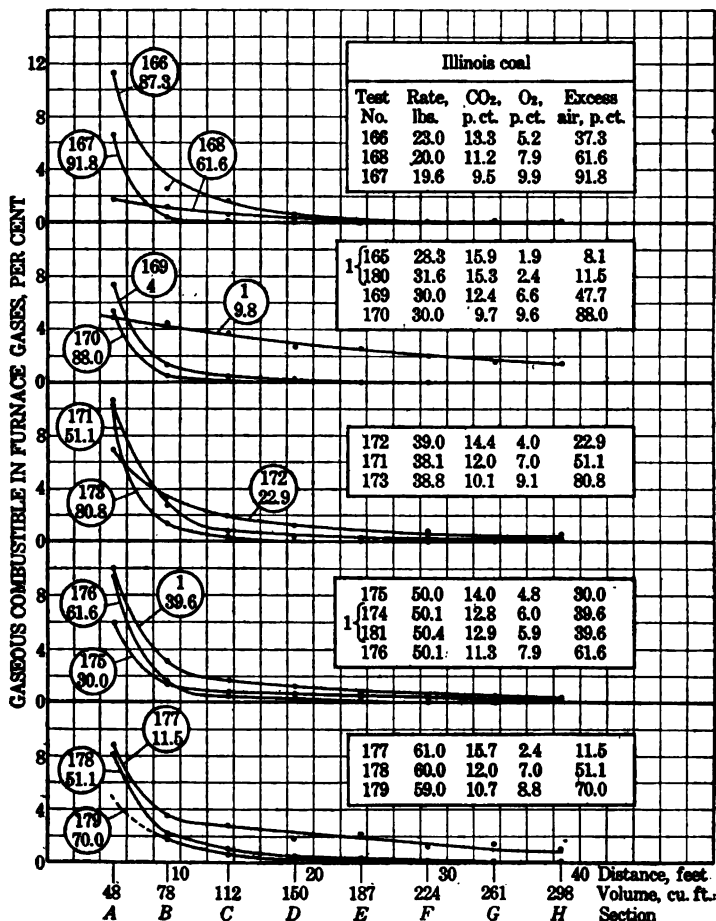


FIGURE 18.—Relation between the percentage of gaseous combustible in furnace gases and the length and volume of combustion space, as shown by tests made with Illinois coal at rates of combustion of approximately 20, 30, 40, 50, and 60 pounds.

collected 1 foot from the surface of the fuel bed is too high in free oxygen and too low in total combustible content. This statement is confirmed by the results shown in figures 14 to 18, inclusive; the points giving the percentage of combustible 1 foot above the fuel bed appear to be all too low, in fact some of them are considerably lower than the points representing samples taken at section A. In drawing the curves the points representing samples taken 1 foot

from the fuel bed and those for samples at section A have been given equal weight and the line passes about midway between the two points. Perhaps the curves should have been passed nearer to the points representing samples from section A.

Although the coal was fed to the magazines at a uniform rate throughout the entire length of a test, the rate of combustion during the sampling period may have been either lower or higher than that shown by the average for the test. Variation in the rate of combustion is apt to occur even with the most careful control of the fire. All the four coals tested tended to cake and to accumulate on the portion of the grate near the magazines, while farther away wide cracks formed in the fuel bed and bare spots on the grate. Consequently the caked coal had to be frequently pulled down over the cracks and bare spots by the furnace attendant. Under such conditions it is possible for the instantaneous rate of combustion to vary 25 per cent below or above the average rate for the entire test.

Comparison of the different groups of curves in figures 14 to 18, inclusive, show that with higher rates of firing more combustion space is needed to obtain nearly complete combustion than when the rate of firing is low. These figures show this relation only in a general way, because the tests with different rates of firing were not run with absolutely the same excess of air. The air supply was the most difficult factor to control. In plotting figures 28 to 30 (pages 73 to 75), some points were interpolated in order to reduce the effect of variation in air supply.

UNDEVELOPED HEAT OF FURNACE GASES ALONG THEIR PATH OF TRAVEL.

The curves of figures 14 to 18, inclusive, are somewhat misleading. The low percentage of combustible in the furnace gases when the excess of air is large is not entirely a result of improved combustion, but to some extent is due to the dilution of the gases with air. It is evident that with a large excess of air, the percentage of combustible gases is less, even if there be no improvement in the combustion. Therefore, when the air supply is varied the percentage of combustible gases does not reliably indicate the completeness of combustion. To eliminate the effect of this dilution with air, the content of gaseous combustible was expressed as a percentage of the calorific value of the coal; that is, the amount of unburned combustible gases represents so many per cent of the total heat value of the coal. In making this change each constituent of the combustible gases was given its heat value. The unsaturated hydrocarbons were considered as C_2H_4 . The following is the general formula used in the calculation:

$$H = \left(\frac{CO \times 12 \times 10,100 + CH_4 \times 16 \times 23,500 + H_2 \times 62,000 + C_2H_4 \times 28 \times 21,300}{12(CO_2 + CO + CH_4 + 2C_2H_4) \times \text{heat of moisture and ash free coal}} \right) \times$$

per cent of carbon in moisture and ash free coal.

In the formula H is the undeveloped heat of the combustible gases in the furnace, expressed as a percentage of the total heat value (B. t. u. per pound) in the coal, and CO_2 , CO , CH_4 , H_2 , and C_2H_4 are the percentages of the respective gases as determined by volumetric analysis. With Pittsburgh coal this formula reduces to the following simpler form:

$$H = \frac{58\text{CO} + 179\text{CH}_4 + 59\text{H}_2 + 285\text{C}_2\text{H}_4}{\text{CO}_2 + \text{CO} + \text{CH}_4 + 2\text{C}_2\text{H}_4}$$

The heat values thus obtained are only approximate, because the equation does not take into account the carbon rising from the fuel bed in the form of soot and tar, but are within a few per cent of the correct value. The maximum error is probably not over 5 per cent. To get the correct value for the undeveloped heat, both the numerator and the denominator of the equation should contain the soot and tar terms. To incorporate these two terms in the equation would be difficult, because the exact composition of the tar and the soot and their heating values are not known. Furthermore, both the gases and the tar and soot would have to be reduced to a common basis of percentage by weight, and it is difficult to obtain the soot and tar sample accurately. As the equation stands H is too high and approaches the correct value as the tar and soot disappear from the furnace gases.

The values of H determined by the equation were plotted as ordinates with the length and volume of combustion space as abscissas. The results so plotted are given in figures 19 to 23, inclusive, the tests being arranged in groups similar to those in figures 14 to 18. Each group contains the results of tests with the same coal at the same rate of combustion, but different amounts of excess air. To avoid crowding in the illustrations, when there is more than one test with the same excess of air, the averages of these tests are plotted instead of each test being plotted separately. The test number and the percentage of excess of air are shown within the small circles attached to each curve.

The curves are similar in shape to those in figures 14 to 18, which are plotted on the percentage of combustible gas as ordinates, with the exception that the rise on the left is steeper. As mentioned in previous paragraphs the points at the left are too high on account of the large amount of tar and soot carried by the gases near the surface of the fuel bed.

It is interesting to note that at the surface of the fuel bed the combustible gases represent 35 to 65 per cent of the total heat value of the coal. This means that under ordinary operation of the side-fed furnace about one half of the total heat in the coal is developed in the fuel bed, the other half being developed in the combustion space. Among other factors, it depends upon the size of the com-

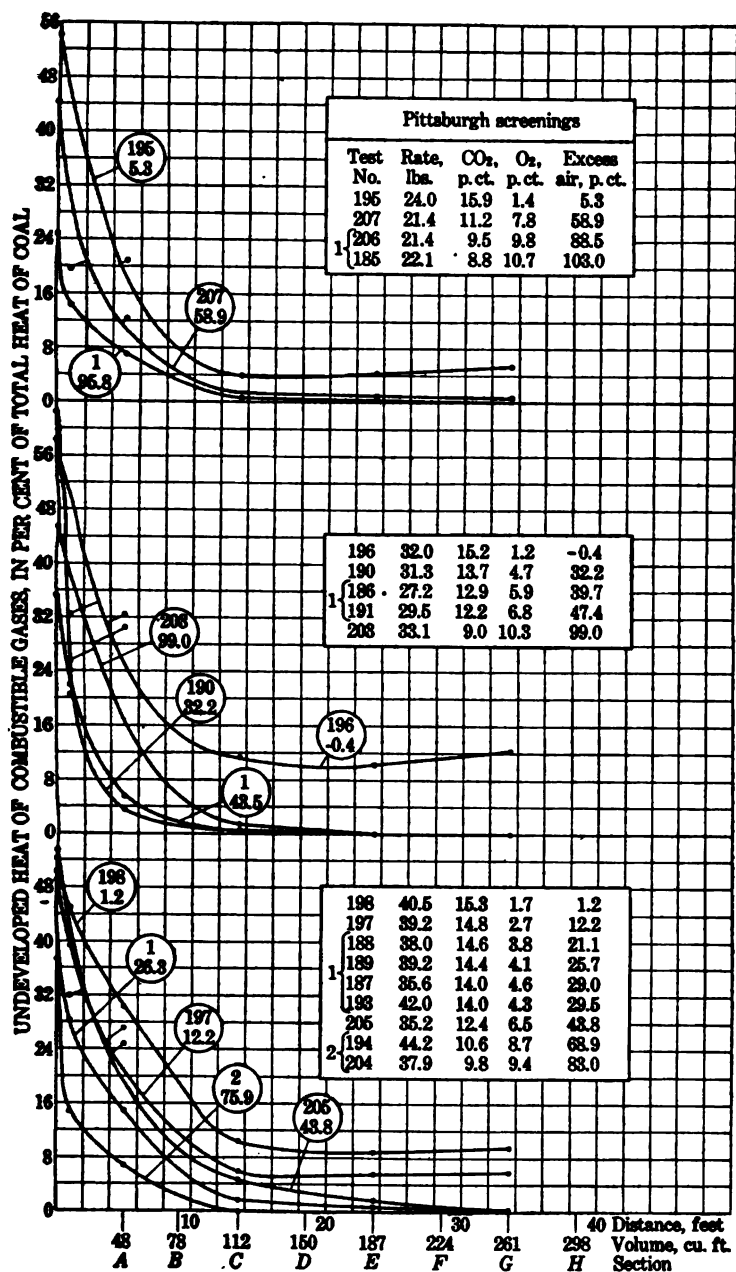


FIGURE 19.—Relation between the completeness of combustion and the length and volume of combustion space, as shown by tests made with Pittsburgh screenings at rates of combustion of approximately 20, 30, and 40 pounds.

bustion space how much of the 50 per cent of heat left in the combustible rising from the fuel bed is developed. The curves of figures 19 to 23 show this relation of completeness of combustion to the length and volume of the combustion space, and also the effect of the excess of air and the rate of firing on the completeness of combustion at the various sections of the combustion space.

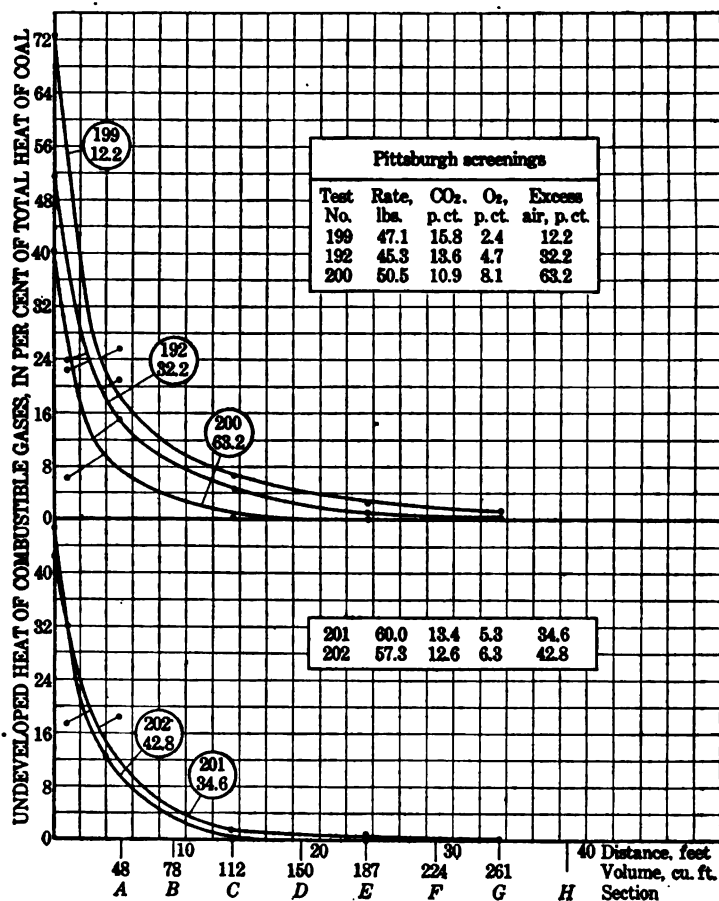


FIGURE 20.—Relation between the completeness of combustion and the length and volume of combustion space, as shown by tests made with Pittsburgh screenings at rates of combustion of approximately 50 and 60 pounds.

In general, the curves of figures 19 to 23 show that when the excess of air is increased the combustion requires less space to become nearly complete. A few tests do not seem to conform to this principle, and their curves fall out of place. There may be several reasons for this discrepancy, the three principal ones being those already stated on pages 56 and 57 in connection with figures 14 to 18.

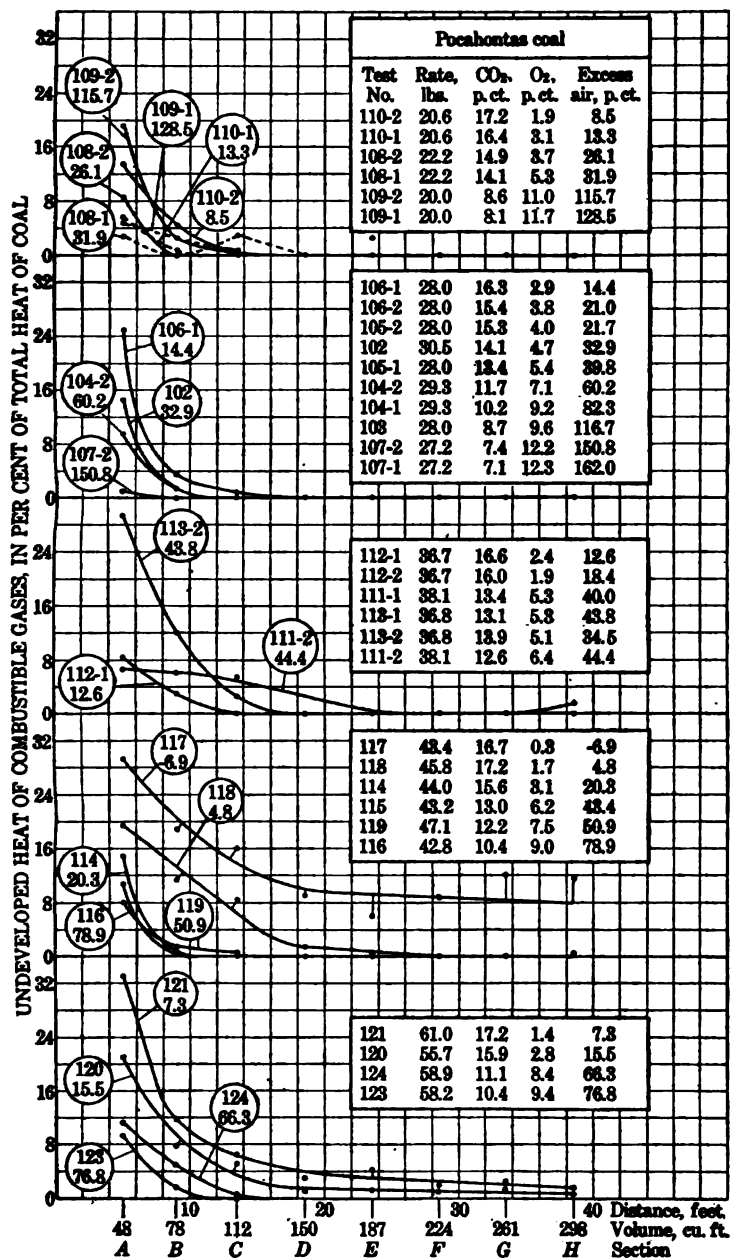


FIGURE 21.—Relation between the completeness of combustion and the length and volume of combustion space, as shown by tests made with Pocahontas run-of-mine coal at rates of combustion of approximately 20, 20.6, 22.2, 23.0, 27.2, 30.5, 38.1, 42.8, 43.2, 47.1, 50.9, 55.7, 58.9, and 61.0 pounds.

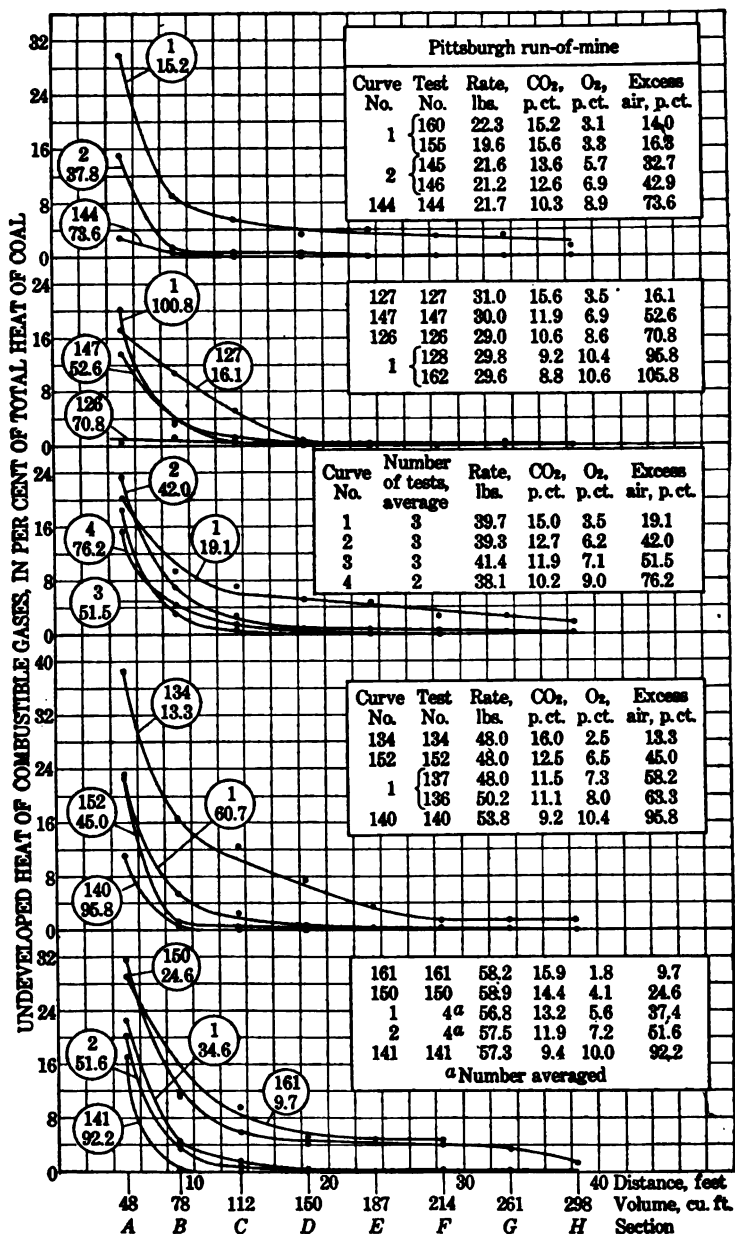


FIGURE 22.—Relation between the completeness of combustion and the length and volume of combustion space, as shown by tests made with Pittsburgh run-of-mine coal at rates of combustion of approximately 20, 30, 40, 50, and 60 pounds.

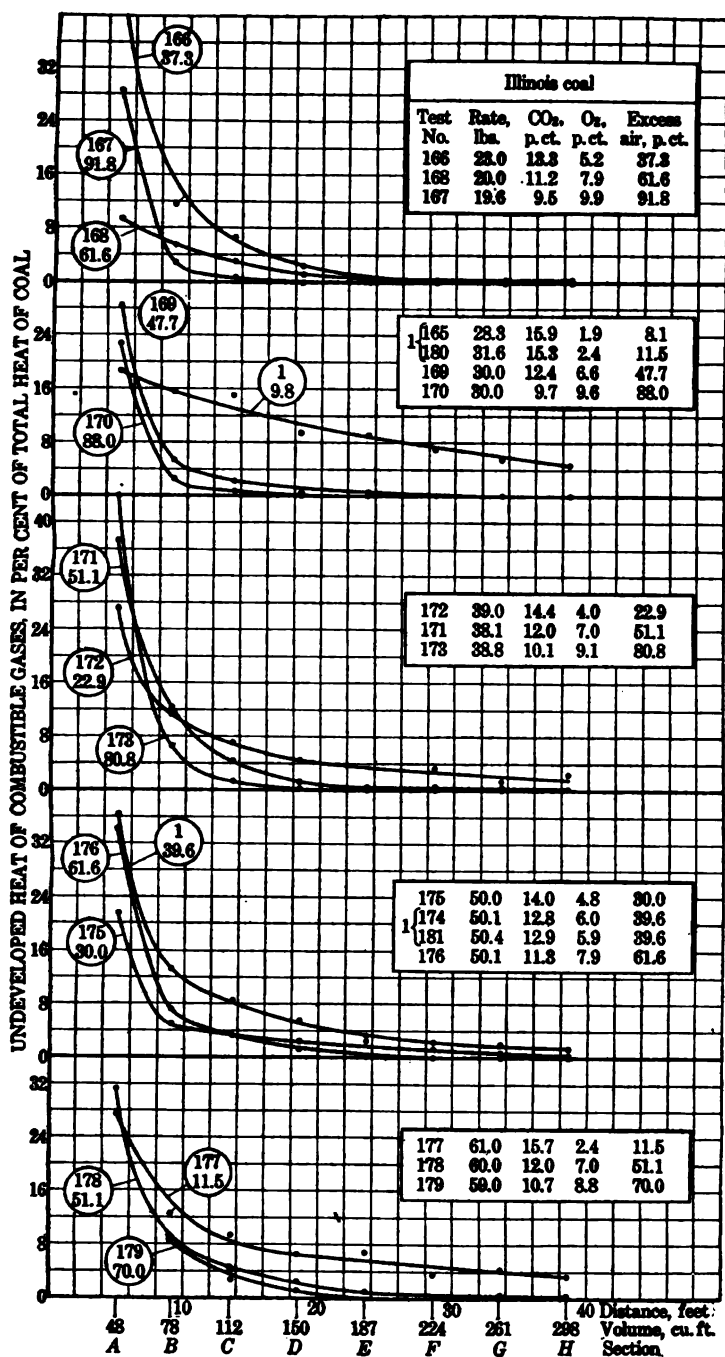


FIGURE 22.—Relation between the completeness of combustion and the length and volume of combustion space, as shown by tests made with Illinois run-of-mine coal at rates of combustion of approximately 20, 30, 40, 50, and 60 pounds.

EFFECT OF THE PERCENTAGE OF EXCESS AIR AND THE RATE OF FIRING ON COMPLETENESS OF COMBUSTION.

The effect of the percentage of excess air on the completeness of combustion is shown better in figures 24 to 27. In these figures the undeveloped heat represented by the combustible gases at each cross section is plotted as ordinates with the percentage of excess of air as abscissas. Each figure represents one coal, and each group of points one section of the combustion chamber. The rate of combustion is indicated by the size or shape of the plotting points. The curves thus obtained show that at any of the cross sections beyond section A the percentage of the undeveloped heat increases as the excess of air becomes smaller. This relation is rather definite, although there are not enough tests with each rate of combustion to give a smooth individual curve. However, the points representing tests with rates of combustion of 40, 50, and 60 pounds invariably fall higher than the points representing rates of 20 and 30 pounds. Thus by dividing each series of tests into two groups, one including all tests with rates of 40, 50, and 60, and the other with rates of 20 and 30 pounds, a sufficient number of tests is obtained to draw two smooth curves, one representing approximately a rate of 50 and the other a rate of 25 pounds. In the series of tests with the Pittsburgh screenings even this grouping does not give enough points for the drawing of two reliable curves.

From the curves of figures 24 to 27, four sets of curves were obtained, showing the completeness of combustion along the path of gases, for the two average rates of firing and the four constant excesses of air. Figure 28 shows a set of these curves representing the series of tests with the Pittsburgh run-of-mine coal. The points used for obtaining these curves are the intersection points of the vertical lines representing constant amounts of excess air with the curves of the two average rates of combustion in figures 24 to 27. The curves themselves have little significance, hence only one set is presented, and were prepared only for compiling the more significant curves shown in figures 29 and 30.

SIZE OF COMBUSTION SPACE REQUIRED FOR ANY DESIRED COMPLETENESS OF COMBUSTION, RATE OF FIRING, AND EXCESS OF AIR.

The relation between the required size of combustion space and any given rate of firing, excess of air and the completeness of combustion is given in figures 29 and 30. Each of these figures gives this relation for two of the four coals tested. The curves to the left of the heavy vertical line dividing the figure into two equal parts represent Illinois coal and those on the right, Pocahontas coal. Each half of the figure shows seven groups of curves, each group representing one degree of completeness of combustion, and each

curve in any one group one percentage of excess of air. In reality each group is a complete figure in itself, having for ordinates rates

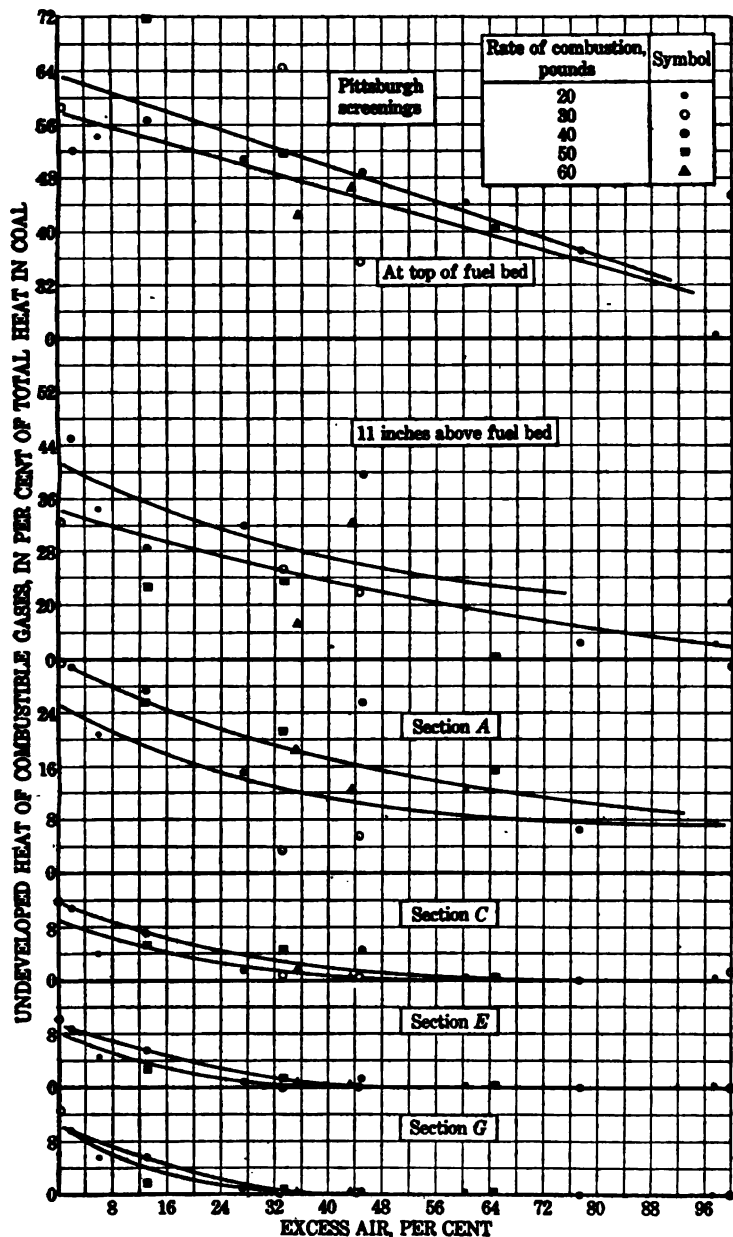


FIGURE 24.—Effect of the percentage of excess air on the completeness of combustion at various cross sections of the long combustion chamber, as shown by tests made with Pittsburgh screenings. The different rates of combustion are indicated by the size and shape of the points.

of firing ranging from 20 to 60 pounds, and for abscissa the size of combustion space, the latter being given in four scales at the foot

of the figure. Reading from top to bottom, the first scale gives the combustion space in cubic feet; the second scale gives the ratio of the combustion space to the area of grate, or the number of cubic feet of combustion space to each square foot of grate area; the third gives the average length of the path of gas travel; and the fourth

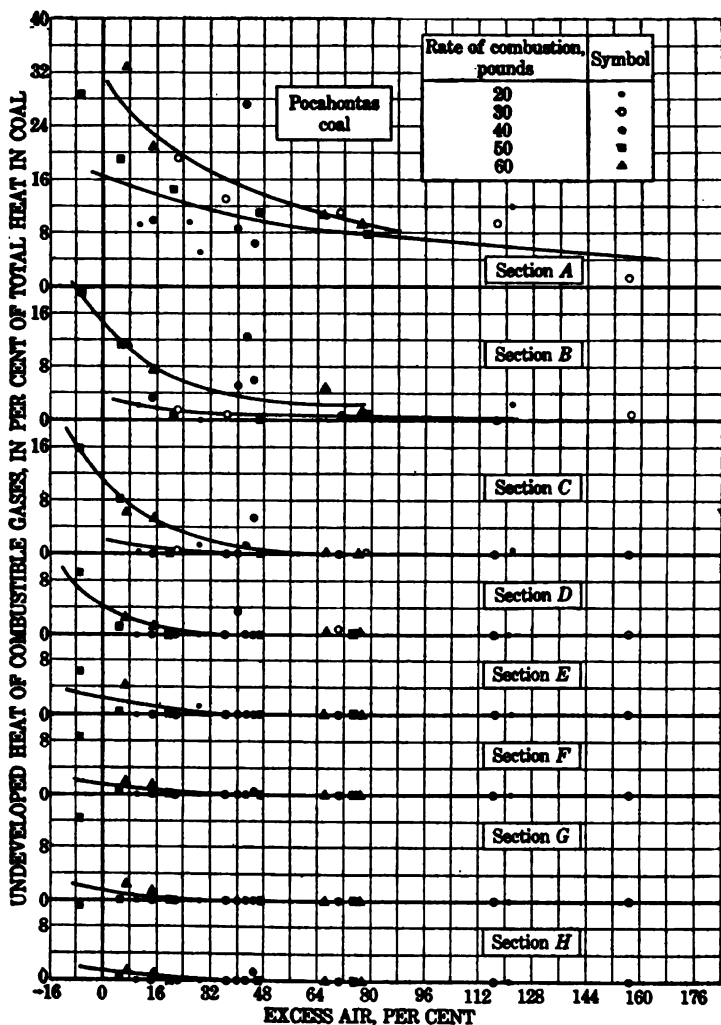


FIGURE 25.—Effect of the percentage of excess air on the completeness of combustion at various cross sections of the long combustion chamber, as shown by tests made with Pocahontas coal. The different rates of combustion are indicated by the size and shape of the points.

gives the sections at which gas samples in the long combustion chamber were collected.

The curves are drawn as a straight line for the reason that only two points were determined for each curve, the two points being obtained by averaging and partly by interpolation. The curves

are approximate within perhaps 10 to 20 per cent. If it were possible to determine more than two points accurately the curve would very

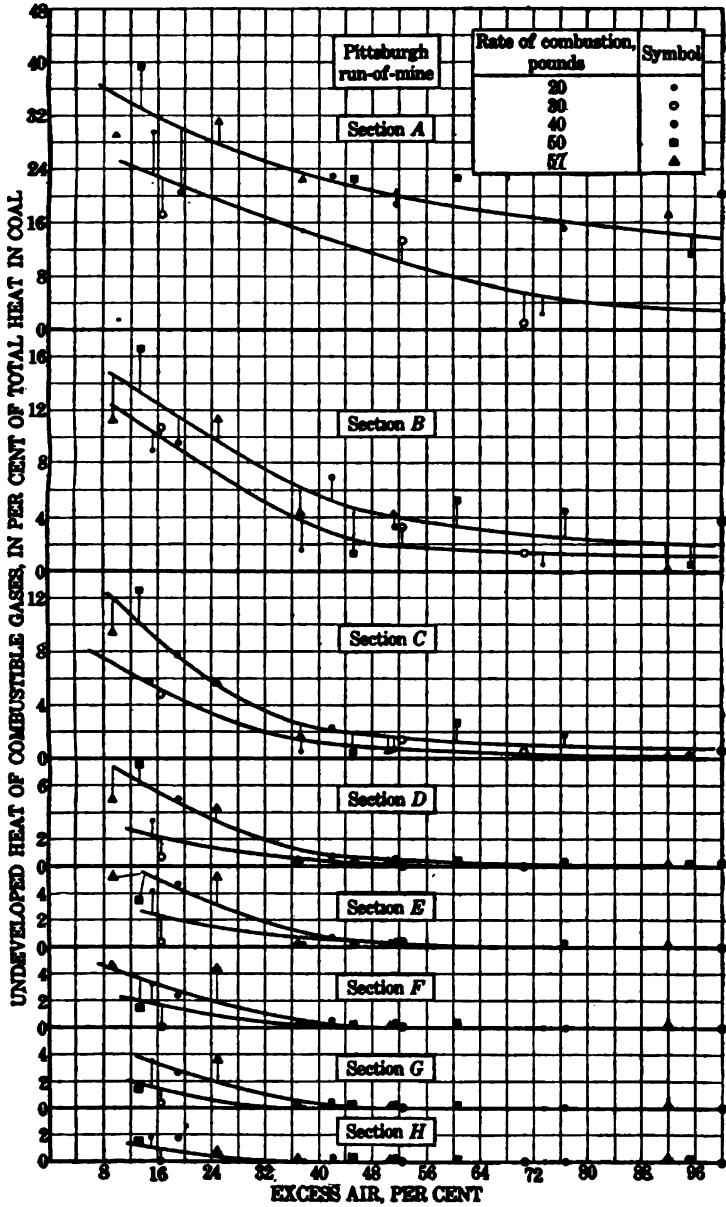


FIGURE 28.—Effect of the percentage of excess air on the completeness of combustion at various cross sections of the long combustion chamber, as shown by tests made with Pittsburgh run-of-mine coal. Different rates of combustion are indicated by differently shaped points.

likely approximate the shape of the letter S, similar to that shown by the dotted curves in the middle of figure 29. It is believed,

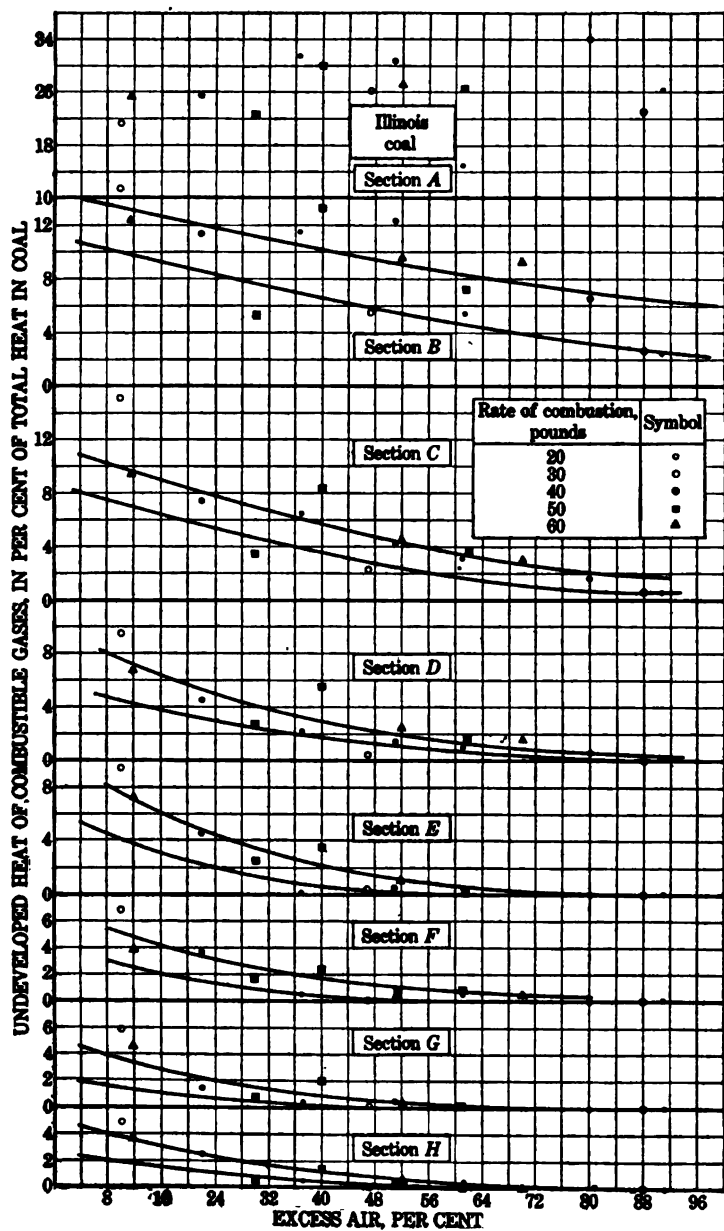


FIGURE 27.—Effect of the percentage of excess air on the completeness of combustion at various cross sections of the long combustion chamber, as shown by tests made with Illinois coal. Different rates of combustion are indicated by differently shaped points.

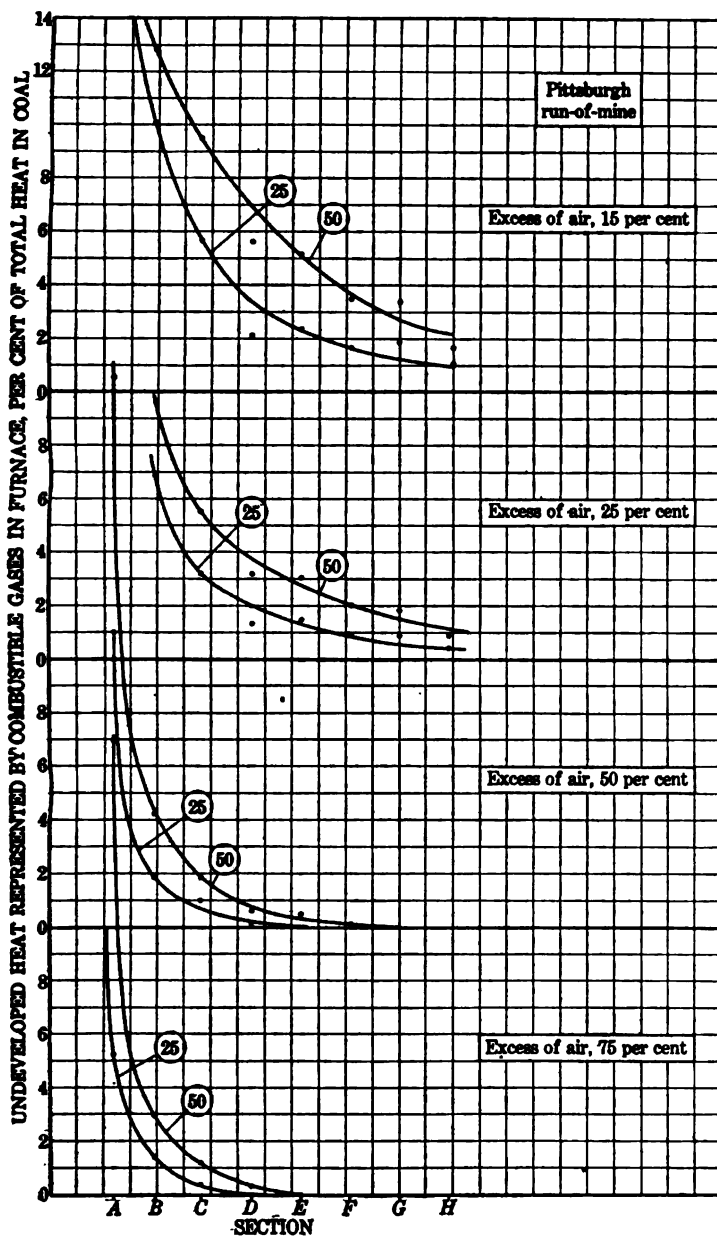


FIGURE 28.—Completeness of combustion along the path of the gases with four percentages of excess of air and average rates of combustion of 25 and 50 pounds, as shown by tests made with Pittsburgh run-of-mine coal.

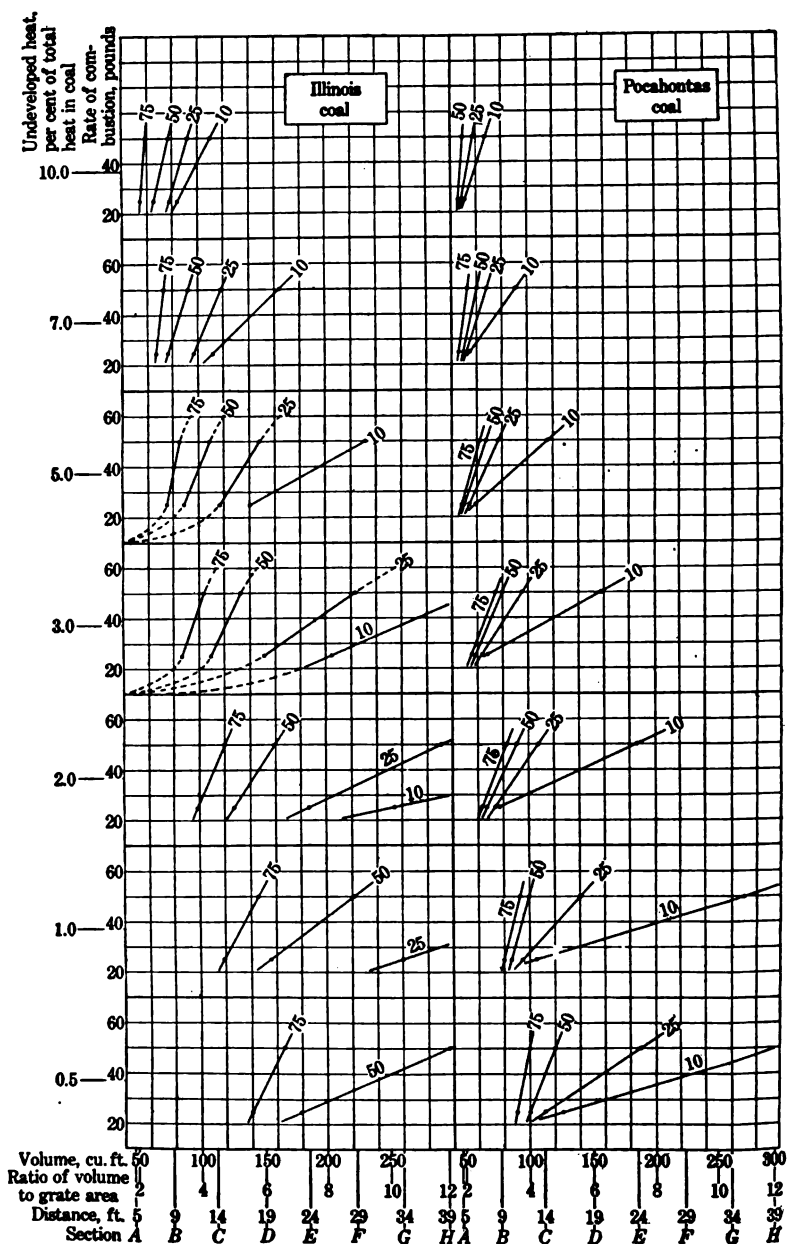


FIGURE 29.—Results of tests with Illinois and Pocahontas coal showing relation between the required size of combustion space, a given completeness of combustion, the rate of firing, and the percentage of excess air. Curves on the left are for Illinois coal; those on the right for Pocahontas coal. Figure on each curve indicates percentage of excess air; figures at left margin opposite each group of curves indicates percentage of undeveloped heat and rate of combustion. Figures at the bottom indicate volumes of combustion space, ratio of volume of combustion space to grate area, length of path of gas travel, and sections at which gas samples were collected in combustion chamber.

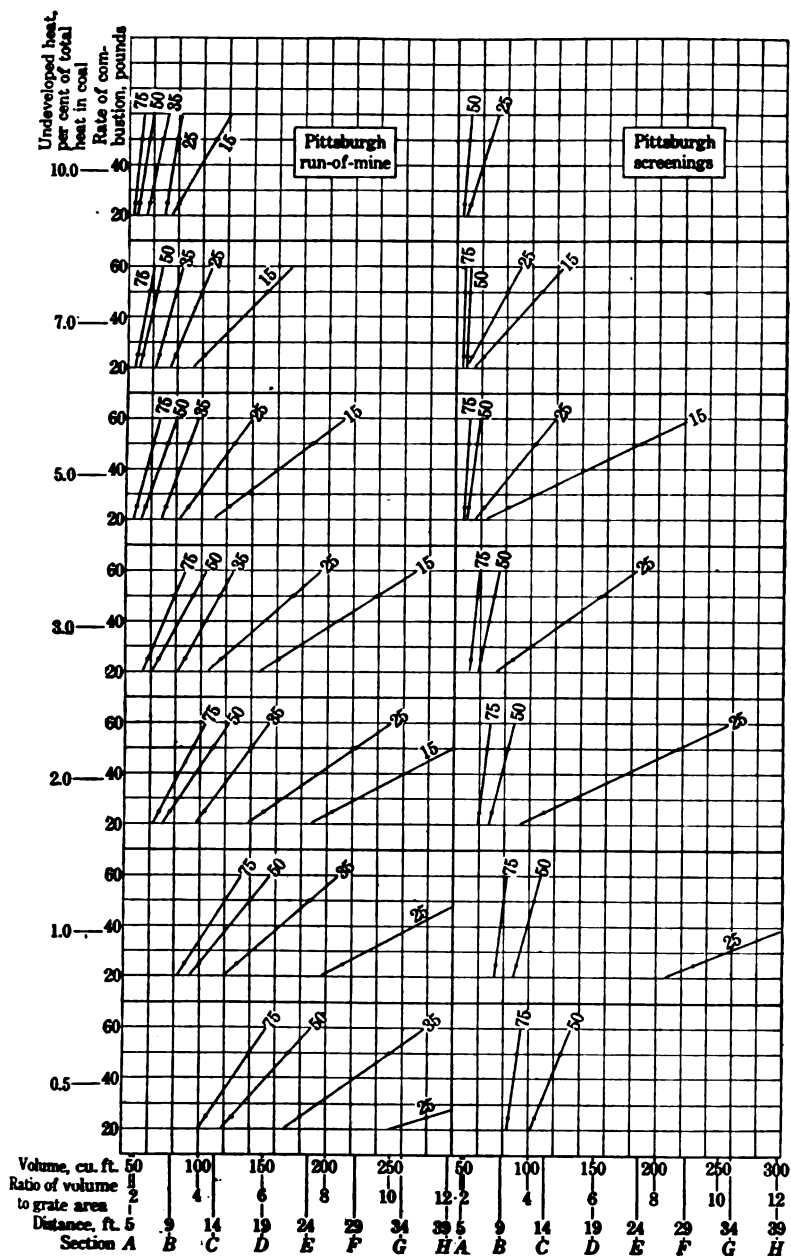


FIGURE 30.—Results of tests with Pittsburgh run-of-mine and Pittsburgh screenings showing relation between the required size of combustion space, a given completeness of combustion, the rate of firing, and the percentage of excess air. Curves on the left are for run-of-mine coal; those on the right, for screenings. Figure on each curve indicates percentage of excess air. Figures at left margin opposite each group of curves indicate percentage of undeveloped heat and rate of combustion. Figures at the bottom indicate volume of combustion space, ratio of volume of combustion space to grate area, length of path of gas travel, and sections at which gas samples were collected in combustion chamber.

however, that between the two points the curves are very nearly a straight line, and inasmuch as the two points cover the rates of combustion used in small and large power stations, the curves have considerable practical value in furnace design for the determination of the size of combustion space when other conditions are given. The curves for the first three coals are more accurate than those obtained with the Pittsburgh screenings, because with the latter not enough tests have been made to insure reliable average points. It is well to state that the series of tests with the Pittsburgh screenings was run mainly with the object of studying the combustion immediately over the fuel bed, and therefore efforts were made to obtain data for that purpose.

PRACTICAL APPLICATION OF RESULTS.

The curves of figures 29 and 30 can be used for determining the size of the combustion space required for given conditions, in the following manner:

Suppose that it is desired to design a furnace that will burn Illinois coal at the rate of 40 pounds per square foot of grate per hour with 50 per cent excess of air and with an incomplete combustion of only 2 per cent of the heat in the coal as fired. For the solution of this problem the left half of figure 29 can be used and the group of curves designated by 2 per cent (undeveloped heat) at the left margin. From the intersection point of the horizontal line of 40 pounds of rate of combustion with the curve of 50 per cent excess of air, a vertical line is followed to the bottom of the figure, where in the second scale the size of the combustion space is found to be 5.8 cubic feet to every square foot of grate. The third scale indicates that the length of gas travel for this condition should be about 18 feet. The first scale at the bottom indicates that with the experimental furnace about 145 cubic feet of combustion space was needed to satisfy the given condition, the space extending to within one foot of section D, as indicated by the lowest scale.

If Pocohontas coal is to be burned under the same condition, the required size of the combustion space is obtained from the group of curves, designated by 2 per cent (undeveloped heat), in the right half of the same figure. From the intersection point of the horizontal line of rate of combustion of 40 pounds with the curve of 50 per cent of excess of air the vertical line is followed to the bottom of the figure, where the second scale indicates that about 3.2 cubic feet of combustion space is needed for every square foot of grate area, and that the length of the gas path should be about 10 feet.

When Pittsburgh run-of-mine coal is to be burned, it is found in the same manner from the left half of figure 30 that the same results can

be obtained with a volume to grate ratio of about 3.9 and an average length of gas travel of about 12 feet.

Thus the three coals, Pocahontas, Pittsburgh, and Illinois coal, require 3.2, 3.9, and 5.8 cubic feet of space per square foot of grate respectively, to burn 40 pounds of coal per square foot of grate per hour, with 50 per cent excess of air and incomplete combustion of 2 per cent of the total heat in the coal fired.

According to the right half of figure 30, when burning Pittsburgh screenings only about 3.1 cubic feet of combustion space is required per square foot of grate to burn the coal with the same results. This is about the same combustion space as that required to burn the Pocahontas coal. The results of the tests with Pittsburgh screenings, however, are not comparable with those obtained with the other three coals because with the Pittsburgh screenings the mixing of the gases was much better. As stated on page 76 the series of tests with the Pittsburgh screenings was undertaken in order to study the combustion immediately over the fuel bed. For that purpose nine sampling tubes were inserted through the arch into the space over the fuel bed. The arch was a double one with an air space between its two parts, as shown in figure 3. This air space was connected with that supplying air to the tuyères, so that the air between the two parts of the arch was under the same pressure as the air entering the tuyères. The holes in the arch for the insertion of the sampling tubes were unavoidably made somewhat larger than the tubes, with the result that air leaked around the sampling tube from the air space between the two parts of the arch into the furnace. Leakage through the outer arch was prevented by packing asbestos rope around the tube, but there was no effective way of preventing the leakage in the inner arch. As a result of this leakage there were nine small streams of air injected into the furnace and evenly distributed over the area of the grate so that the air mixed readily with the combustible gases, thus making the combustion over the fuel bed more rapid than was the case with the other series of tests where there was no leakage through the arch. The series of tests with the Pittsburgh screenings shows the effect of mixing on the rapidity of combustion.

EFFECT OF LENGTH OF GAS TRAVEL ON THE COMPLETENESS OF COMBUSTION.

When considering the volume of combustion space, it is well to add that the length of the gas travel is probably an important factor. It seems that a long narrow combustion space is more efficient in burning the gases than a short wide one having the same cubical space. In the long narrow space the gases travel with a higher velocity, which promotes mixing and therefore quickens the combustion. In the short wide space the gases remain the same length of time, but

travel slower. On account of this slower movement the gases are less agitated and tend to travel in stratified streams. Therefore there is less mixing and the combustion is slower. It is advisable that in using the data of figures 29 and 30 in designing a furnace the path of the gases should be made nearly as long as in the experimental furnace if practicable.

EFFECT OF COMPOSITION OF COAL ON THE SIZE OF COMBUSTION SPACE REQUIRED.

Coals having different composition require different sizes of combustion space. The chemical characteristics that are most likely to affect the size of the combustion space required for a given set of conditions are the quantity and quality of the volatile matter. The quantity of the volatile matter is shown by its percentage as determined by the proximate analysis. The quality of the volatile matter is indicated approximately by the ratio of volatile carbon to available hydrogen, and probably also by the oxygen content of the coal on a moisture and ash free basis. The principal chemical characteristics of the three coals tested are given in Table 4 following. Items 1, 6, and 7 of the table give the three mentioned factors which are most likely to affect the burning qualities of any coal. Roughly speaking, the characteristics of the three coals as given in items 1 and 6 increase in about equal steps from Pocahontas to Pittsburgh, and from Pittsburgh to Illinois coal. On the other hand, the difference in the oxygen content between the Pittsburgh and the Illinois coal is much larger than it is between the Pocahontas and the Pittsburgh coal.

TABLE 4.—*Chemical characteristics of the three coals tested.*

Item No.	Item.	Pocahontas coal.	Pittsburgh coal.	Illinois coal.
1	Volatile matter in moisture and ash free coal..... per cent..	18.05	34.77	46.52
2	Fixed carbon in moisture and ash free coal.....do....	81.95	65.23	53.48
3	Carbon in moisture and ash free coal.....do....	90.50	85.7	79.7
4	Volatile carbon in moisture and ash free coal.....do....	8.55	20.47	26.22
5	Available hydrogen in moisture and ash free coal.....do....	3.96	4.70	3.96
6	Ratio of volatile carbon to available hydrogen.....do....	2.16	4.35	6.6
7	Oxygen in moisture and ash free coal.....do....	3.32	5.59	10.93
8	Nitrogen in moisture and ash free coal.....do....	1.19	1.73	1.70
9	Percentage of moisture accompanying 100 per cent of moisture and ash free coal.....	2.53	2.88	22.07
10	Volatile matter times the ratio of volatile carbon to available hydrogen (product of items 1 and 6).....per cent..	39.0	151	307
11	Ratio of oxygen to total carbon, in moisture and ash free coal..	.0367	.0652	.137
12	Total moisture in furnace per pound of coal reduced to moisture and ash free basis.....pounds..	.409	.501	.700

THE EFFECT OF THE QUANTITY OF THE VOLATILE MATTER.

Table 5 gives the size of the required combustion space for the three coals and several sets of conditions indicated by columns 1, 2, and 3 of the table. Examination of the values in columns 4, 5, and 6

shows that the size of the combustion space does not increase in direct proportion to the percentage of the volatile matter in the coal. The increase in the combustion space from Pittsburgh to Illinois coal is much larger than the increase from Pocahontas to Pittsburgh coal. Roughly speaking, under the same conditions Pittsburgh coal requires about 20 per cent larger combustion space than Pocahontas coal, whereas Illinois coal requires about 40 per cent larger combustion space than Pittsburgh coal. That the size of the combustion space does not increase in direct proportion to the percentage of volatile matter in the coal is shown graphically by the upper curves of figure 31. If the relation of the size of combustion space to the percentage of volatile matter were a direct proportion, the relation would be represented by a straight line. The figure shows that the curves are far from straight lines, and become more and more curved as conditions of less complete combustion are considered and the combustion space becomes smaller. However, in the opposite direction toward complete combustion the curves seem to approach a straight line.

That the size of the required combustion space under ordinary degrees of completeness of combustion does not vary in direct proportion as the quantity of the volatile matter, even if the quality of the latter remains constant, can be deduced from Table 5 by comparing the two rates of combustion of the same coal. Thus, when the rate of combustion is doubled the quantity of the volatile matter distilled per unit of time is doubled. However, to burn this double quantity of volatile matter with the same excess of air to the same completeness, the combustion space is increased only about 20 per cent. Table 5 follows.

TABLE 5.—Size of combustion space required for the three coals when burned under conditions given in columns 1, 2, and 3.

Completeness of combustion, per cent of undeveloped heat.	Rate of combustion, pounds per square foot of grate per hour.	Excess of air (per cent).	Cubic feet of combustion space per square foot of grate area.		
			Pocahontas.	Pittsburgh.	Illinois.
1	2	3	4	5	6
5	50	50	2.7	2.9	4.3
3	50	50	3.2	3.7	5.3
2	50	50	3.6	4.4	6.3
1	50	50	4.0	5.6	8.9
0.5	50	50	4.8	6.8	11.9
5	25	50	2.0	2.2	3.5
3	25	50	2.3	2.7	4.35
2	25	50	2.7	3.1	5.1
1	25	50	3.4	4.0	6.2
0.5	25	50	4.0	5.0	7.1

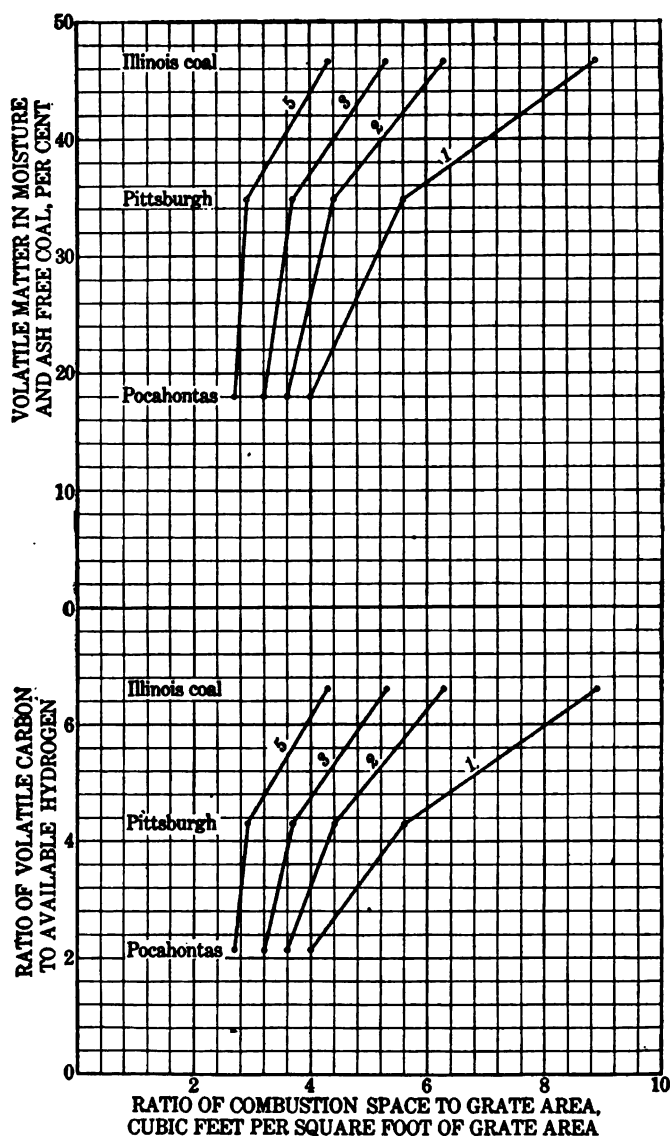


FIGURE 31.—Relation between volatile matter of coal and size of combustion space required. The curves are plotted from the first set of data in Table 5. Upper group shows relation between the size of required combustion space and the percentage of volatile matter; lower group shows relation between the size of required combustion space and the quality of the volatile matter as shown by the ratio of volatile carbon to available hydrogen. Figure near each curve indicates percentage of heat not developed, owing to incomplete combustion. In all tests the rate of combustion is 50 pounds and the excess of air 50 per cent.

THE EFFECT OF THE QUALITY OF THE VOLATILE MATTER.

The quality of the volatile combustible, as far as the ease of burning it is concerned, is perhaps best expressed by item 6 in Table 4, showing the ratio of volatile carbon to available hydrogen. These values were obtained by dividing the volatile carbon by the available hydrogen, and are probably fair indicators of the burning qualities of the coals. The amount of volatile carbon was computed by subtracting the amount of fixed carbon from that of the total carbon. The available hydrogen is equal to the hydrogen content on a moisture and ash free basis minus one-eighth of the oxygen content. The ratio shows that the volatile matter of the Pittsburgh coal contains nearly twice as much carbon and that of the Illinois coal three times as much carbon as the Pocahontas coal. These ratios indicate the probability that in burning Pocahontas coal the volatile combustible is distilled off mostly as light gases which are easily burned in the diluted furnace atmosphere, whereas in burning Illinois coal the volatile combustible leaves the fuel bed mostly as heavy hydrocarbons in the form of tars, which in the diluted oxygen of the furnace atmosphere are first decomposed into lighter hydrocarbons and carbon, the latter being precipitated as soot. This mixture of soot, tar, and gases burns slowly and requires a large combustion space for its complete combustion.

The effect of high carbon content of the carbon-hydrogen compounds on the rapidity of combustion is discussed more fully on pages 118 and 119. Here it may be repeated that in general the higher the carbon content in the carbon-hydrogen compounds the more time is required for their combustion. Therefore it may be expected that as the ratio of volatile carbon to available hydrogen increases the size of the combustion space required for a given degree of completeness also increases. The data in Table 5 and the lower set of curves of figure 31 show such an increase, but the increase is not in direct proportion to the ratio, especially with conditions of less complete combustion. As perfect combustion is approached the curve representing this relation seems to approach more nearly a straight line.

Inasmuch as both the quantity and the quality of the volatile matter exert an influence in the same direction on the burning property of a coal, the size of the combustion space required for a given completeness of combustion under a given set of conditions should vary in some rather pronounced way as the product of the quantity and the quality of the volatile matter. Such product of the percentage of volatile matter in moisture and ash free coal and the ratio of volatile carbon to available hydrogen is given in line 10 of Table 4. In the upper group of curves in figure 32 this product

is plotted against the size of combustion space shown by the first set of data in Table 5. Each curve of the upper group gives the relation

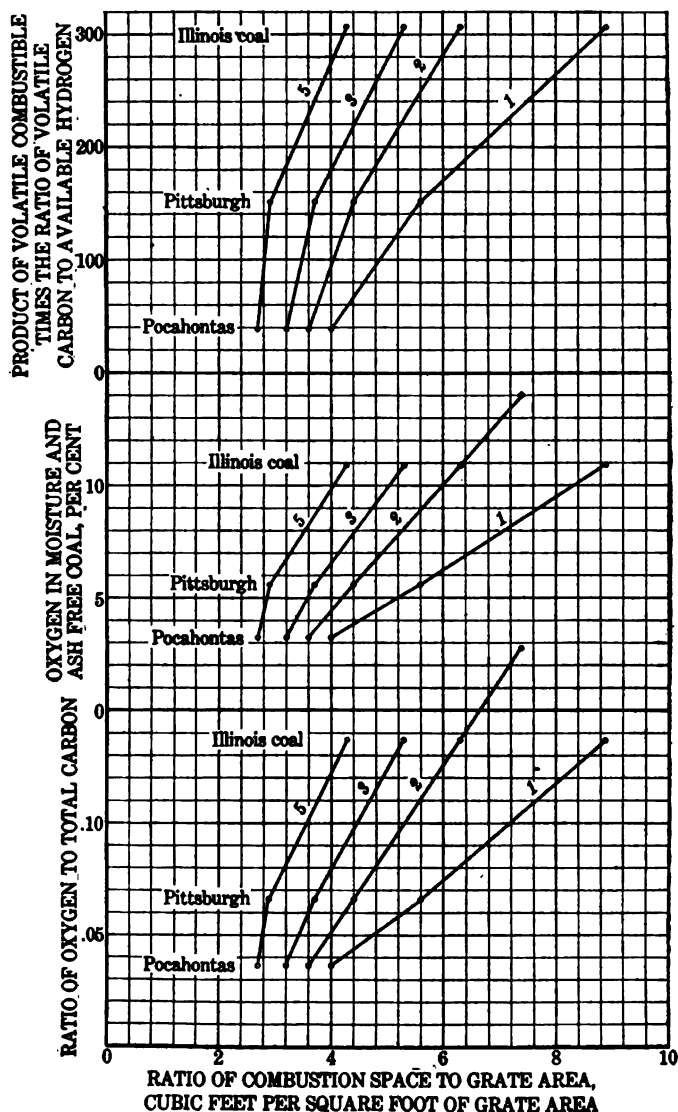


FIGURE 32.—Relation between size of combustion space required and the chemical properties of three coals. The upper group of curves has for its ordinates the product of the percentage of volatile matter times the ratio of volatile carbon to available hydrogen, from line 10, Table 4; the middle group, the percentages of oxygen in "moisture and ash free" coal, from line 7, Table 4; and the lower group, the ratio of total carbon to oxygen in "moisture and ash free" coal, from line 11, Table 4. All three groups have a common abscissa, the ratio of the combustion space to grate area, plotted from the first set of data in Table 5. Figure near each curve indicates percentage of heat not developed, owing to incomplete combustion. In all tests the rate of combustion is 50 pounds and the excess of air 50 per cent.

between the size of combustion space and the product of quantity times the quality of volatile matter for one degree of completeness

of combustion indicated by the figure near each curve. The curves are similar in form to those of figure 31, but are somewhat nearer to being a straight line, particularly the curves for more complete combustion. It may then be said that, in a rough way, when nearly complete combustion is desired the size of combustion space varies directly as the product of the quantity and the quality of the volatile matter as the two are given in items 1 and 6 of Table 4. As the combustion becomes less complete the curve showing the relation between this product and the size of the combustion space is farther from a straight line.

OXYGEN AS AN INDICATOR OF THE BURNING QUALITY OF COALS.

The ratio of the volatile carbon to available hydrogen is only one of probably several factors which may indicate the burning qualities of volatile combustible. Among the other factors the oxygen content in "moisture and ash free" coal suggests itself as the most important one.

In line 7, Table 4, the percentages of oxygen in Pocahontas, Pittsburgh, and Illinois coals are shown to be 3.32, 5.59, and 10.93, respectively. According to these figures, the Illinois coal contains by far the largest percentage of oxygen. At the present state of knowledge of combustion of coal it is difficult to explain why oxygen when contained in the volatile matter should delay its combustion. Nevertheless there are some indications that volatile combustible containing a high percentage of oxygen is harder to burn than one containing little oxygen. Whether the difficulty in burning it is due to the oxygen content or to some other chemical property can not be stated definitely. It is interesting to note that with the three coals tested the combustion space required for the same degree of completeness of combustion is almost directly proportional to the oxygen content in "moisture and ash free" coal. That is, if tests of the three coals run at the same rate of combustion, the same excess of air, and the same completeness of combustion are plotted on the oxygen contents as abscissa and the ratio of combustion space to grate area as ordinate, the curves connecting each group of three points representing the three coals are nearly straight lines. Four of such curves are given in the middle of figure 32. In the absence of more complete data the authors are inclined to believe that this nearly straight line relation between the required combustion space and the oxygen content is purely accidental. However, as some writers^a attribute to the oxygen content in coal considerable importance the authors are presenting this relation for the consideration of those interested in this subject.

^a White, David, The effect of oxygen in coal: Bull. 20, Bureau of Mines, 1911, p. 80.

The authors wish to add that it is generally conceded that the oxygen content shows to a certain measure the age of the coal—that is, it shows what stage the process of carbonization of the original plant substance has reached. On the average the original plant substance from which coal was formed contained about 40 per cent of oxygen and a comparatively low percentage of carbon. As the process of the formation of coal went on the oxygen was gradually decreased and the proportion of the carbon thereby increased. Anthracite coal contains only about 2.5 per cent of oxygen and a very high percentage of carbon. With the progress of carbonization, the percentage of the volatile matter also decreases and its quality undergoes a change which makes it easier to burn. Thus, although the oxygen content itself may not directly affect the combustion qualities of a coal it may be regarded in a way as an indicator of such qualities.

In other words, there is probably a certain chemical property of the coal which affects its combustion qualities, and this chemical property varies with the content of oxygen, although the variation in one is not the cause nor the result of the variation in the other, changes in both being brought about by the same cause, the progress of the carbonization of the plant substance.

Inasmuch as with the progress of carbonization the percentage of oxygen decreases and that of the carbon increases, it would seem that the ratio of the oxygen content to the content of total carbon would be a more direct indicator of the age of coal, and therefore also an indicator of its qualities affecting the ease of burning it in industrial furnaces. The group of curves at the bottom of figure 32 gives the relation between this ratio and the required combustion space. In the figure, this relation is represented by a nearly straight line, which fact indicates that the required combustion space increases nearly directly as the ratio; this is particularly true with the conditions giving nearly complete combustion. These curves have the same shape as those of the middle group of the figure.

That the oxygen content is a good indicator of the stage of the process of formation of coal is shown by figure 33, which is a transcription of a chart compiled by Ralston and published in Technical Paper 93^a as Plate I. In Ralston's chart thousands of analyses of coals and other fuels of the United States are plotted on trilinear coordinates representing the carbon, hydrogen, and oxygen contents in moisture-free, ash-free, nitrogen-free, and sulphur-free coal. That is, the coal is assumed to consist of only carbon, hydrogen, and oxygen, containing none of the oxygen and hydrogen which can be driven away as moisture.

^a Ralston, O. C., Graphic studies of ultimate analyses of coals: Tech. Paper 93, Bureau of Mines, 1915, 41 pp.

In the figure the small triangle represents the trilinear diagram, with each of the three coordinates running from 0 to 100. The horizontal distances are the percentages of carbon, starting at the right and ending with 100 per cent at the left-hand corner. The vertical distances are the percentages of hydrogen, beginning at the bottom of the triangle and ending with 100 per cent at the upper corner. The oxygen begins with the slanting side of the triangle as zero and runs toward the lower right-hand corner as 100 per cent. As all the coals and other fuels fall within the small shaded part of the triangle, this part is shown greatly enlarged, in order to show more in detail the position of the different coals.

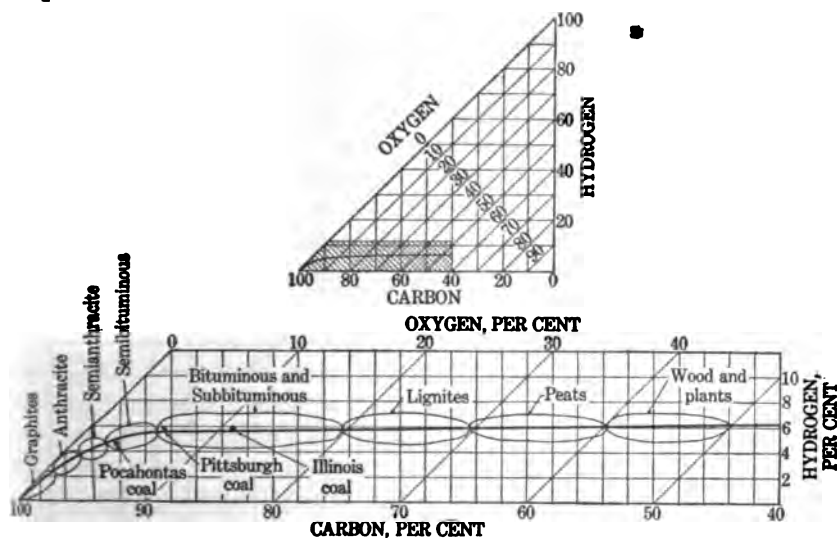


FIGURE 33.—General characteristics of various fuels and their relation as shown by their carbon, hydrogen, and oxygen contents reduced to moisture-free, ash-free, nitrogen-free, and sulphur-free basis, plotted on trilinear coordinates. Shaded area in triangle represents part covered by coals and other fuels. Trapezoid below represents this area greatly enlarged, showing the fields covered by various grades of coals and other fuels. The three black dots show the position of the three coals tested. Compare this figure with figures 31 and 32.

This enlarged part of figure 33 does not contain points representing the individual coals as given in Ralston's chart, but outlines the general grouping of the coals and the fields in which the different fuels fall on the chart. The position of each of the three coals used in tests with the long combustion chamber is indicated on the chart by a black dot designated by the name of the coal. The striking feature of the chart is the fact that, starting with wood, the oxygen content is about 42 per cent, and it gradually decreases through the peats, the lignites, subbituminous, bituminous, semibituminous, and anthracite coal to less than 2.5 per cent.

THE EFFECT OF MOISTURE IN COAL ON COMBUSTION.

Moisture in coal is another chemical characteristic which may affect the combustion. Although in moderate percentages the moisture in furnace gases may be beneficial, in large quantities it may be a detriment and reduce the velocity of combustion. Figure 34, prepared from data by Mellor,^a shows the effect of water vapor on the velocity of combustion of two volumes of CO with one volume of oxygen. The abscissas give the water vapor in per cent by volume. The ordinates are numbers proportional to the velocity of combustion. The curve shows that the maximum velocity of combustion is obtained with about 5 or 6 per cent of moisture. On either side of this maximum

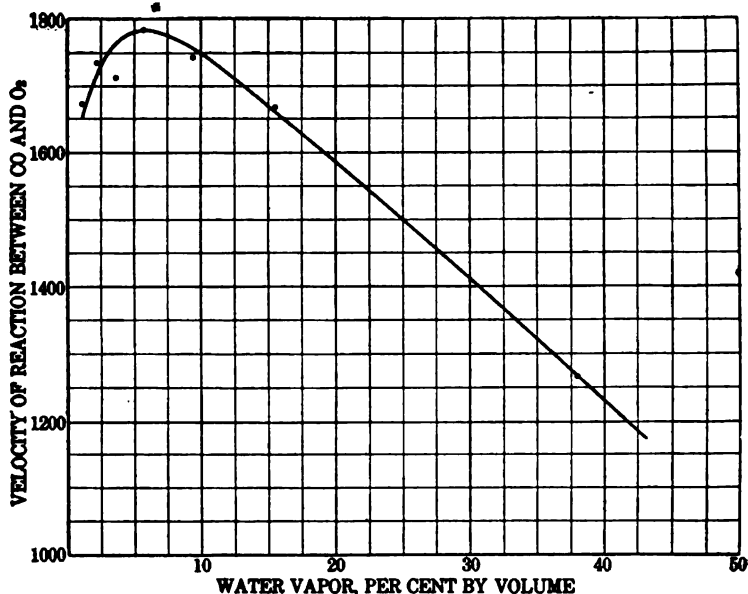


FIGURE 34.—Relation between the percentage of water vapor and the velocity of reaction between two volumes of CO and one volume of O_2 . Curve compiled from data by Mellor.

percentage the velocity of combustion drops rapidly. With no moisture present, the mixture of the two gases would not explode, showing that a small amount of water vapor is necessary for the two gases to unite.

How the presence of different percentages of moisture would affect the combustion of CO with O_2 when the two reacting gases are greatly diluted by CO_2 and N_2 can not be directly deduced from figure 34. It is possible that a curve of similar shape holds true for the combustion of CO under furnace conditions and that with the Illinois coal the furnace gases contain more moisture than the percentage giving the highest velocity of combustion. Under the conditions expressed by curve 3 in figure 31, and using Illinois coal, the percentages of the

^a Mellor, J. W., *Chemical Statics and Dynamics*, 1904, p. 471.

different constituents in the furnace are as follows: CO, 0.6; CO₂, 13.0; O₂, 4.0; H₂O, 7.0; N₂, 75.4.

Thus, although the water vapor constitutes a small percentage of the furnace gases, its volume is considerably greater than the total volume of the two reacting gases CO and O₂. It seems probable, therefore, that moisture in such high quantity retards combustion.

Considering the moisture entirely from the law of mass action, its presence in the furnace is always objectionable, because it reduces the concentration of the reacting gases. Whether it has chemically any retarding effect on the combustion when contained in large quantity can not be definitely stated, as no data on the subject except that given in figure 34 seem to be available.

The nature of combustion and of the coordinates used in figures 31 and 32 are such that none of the relations shown by the five groups of curves can really be represented by a straight line. The straight lines which occur in these figures are merely accidental and are very likely a straight part of a curve or represent only a special case of a general law.

Some of the combustible gases that are burned in the combustion space are not a part of the volatile combustible at all, but are the results of reduction of CO₂ and H₂O in the reducing zone of the fuel bed. It has been shown in experiments described in Technical Paper 137^a that even when coke containing practically no volatile matter is burned, the gases rising from a 6-inch fuel bed contain as much as 18 per cent of CO. Similar results were also obtained with anthracite coal. Inasmuch as the air needed to burn this CO is usually added some distance above the surface of the fuel bed, the gases may have to pass through a considerable volume of combustion space before they are thoroughly mixed with the air and burned. Thus, although these fuels do not have any volatile matter, some combustion space must be provided for mixing the CO with air and burning it. Therefore, in going from the lower grade of coals to Pocahontas coal and coke, the combustion space required does not decrease as fast as the factors indicating the burning properties of the fuel. For this reason a curve representing any set of conditions may appear as a straight line in the region of low-grade coals, but in the low-volatile fuels it has a perceptible curvature. This curvature is always in the direction that makes the combustion space larger.

There is no fuel that will burn almost completely without any combustion space. No matter how much the volatile content of the coal is decreased, the combustion space can not be decreased beyond certain limits except at the expense of completeness of combustion.

^a Kresinger, Henry, Ovitz, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces: Technical Paper 137, Bureau of Mines, 1916, p. 43.

RELATION BETWEEN KIND OF COAL USED AND THE COMPLETENESS OF COMBUSTION.

Table 6 following shows the relation between the kinds of coal and the completeness of combustion for six sets of conditions, given in columns 1, 2, and 3. The degree of completeness of combustion is indicated by the percentage of undeveloped heat. The values in the last three columns show that there is no simple relation. Sometimes the difference in incomplete combustion is greater between Pocahontas and Pittsburgh coal, and sometimes the difference is greater between Pittsburgh and Illinois coal. The degree of the completeness of combustion seems to depend on what part of a curve the three points are placed by the conditions governing the combustion.

TABLE 6.—*Relation between the kind of coal and the completeness of combustion when other conditions are constant.*

Rate of combustion, pounds per square foot of grate per hour.	Excess of air (per cent).	Cubic feet of combustion space per square foot of grate area.	Undeveloped heat, per cent of total heat in coal.		
			Poca-hontas.	Pitts-burgh.	Illinois.
50	75	4.0	0.5	1.7	3.4
50	50	4.8	.5	1.7	4.0
50	25	7.5	.5	2.6	4.0
25	75	3.6	.5	1.0	2.9
25	50	4.0	.5	1.0	4.0
25	25	4.5	.5	2.5	5.2

DETERMINATION OF THE SIZE OF THE COMBUSTION SPACE OF A FURNACE TO BURN A GIVEN COAL.

Figure 32 can be used as a guide to design furnaces for burning given coals. Suppose that it is desired to design a furnace for each of the three coals given in Table 7, the combustion to be complete to within 2 per cent when the coal is burned at the rate of 50 pounds per square foot of grate per hour and with 50 per cent excess of air. The analyses of the coals have been taken at random from Bulletin 23^a. In the table, lines 7, 13, and 14 give the factors indicating the burning qualities of the three coals. By the use of these three factors and figure 32 the size of the combustion space can be approximately determined. Thus with the New Mexico coal the percentage of oxygen given in line 7, Table 7, is 14.02. The point on the prolonged curve marked 2, in the middle set of curves in figure 32, representing this percentage, shows that the size of combustion

^a Breckenridge, L. P., Krelsinger, Henry, and Ray, W. T., *Steaming tests of coals and related investigations*, Sept. 1, 1904, to Dec. 31, 1906: Bull. 23, Bureau of Mines, 1912, 380 pp.

space required for the given condition is 7.4 cubic feet to each square foot of grate. The product of the quantity and the quality of the volatile combustible given in line 13, Table 7, and curve 2 of the top group in figure 32 show that the furnace should have about 6.7 cubic feet of combustion space to each square foot of grate area. The ratio of the oxygen to total carbon given in line 14, Table 7, and curve 2 of the lowest group in figure 32 show that 7.4 cubic feet of combustion space is needed for each square foot of grate. The position of the three points on the diagram is shown by the small circle on the upper end of curve 2 in each group. Thus the ratio of the combustion space to the grate area as given by the three factors varies from 6.7 to 7.4.

Similarly the size of the combustion space of a furnace to burn the Virginia coal under the given conditions is indicated by determining the points representing the three factors of lines 7, 13, and 14 (Table 7) on curve 2 of the respective groups in figure 32. In this instance the three factors indicate that the required combustion space per square foot of grate area should be 4.9 to 5.1 cubic feet.

For the New River, W. Va., coal the three factors indicate that the ratio of the combustion space to the grate area varies from 3.7 to 3.9.

TABLE 7.—Composition of three given coals and factors indicating their burning qualities.

PROXIMATE ANALYSIS OF COAL AS RECEIVED.

Line.	Item.	New Mexico coal.	Virginia coal.	New River (W. Va.) coal.
1	Moisture.....per cent..	11.90	3.37	1.85
2	Volatile.....do.....	37.85	33.75	20.31
3	Fixed carbon.....do.....	41.57	56.41	73.87
4	Ash.....do.....	8.68	6.47	3.95

ULTIMATE ANALYSIS OF MOISTURE AND ASH FREE COAL.

5	Carbon.....per cent..	78.48	84.70	88.50
6	Hydrogen.....do.....	5.52	5.32	5.02
7	Oxygen.....do.....	14.02	7.53	4.28
8	Nitrogen.....do.....	1.28	1.55	1.55
9	Sulphur.....do.....	.70	.90	.65

PROXIMATE ANALYSIS OF MOISTURE AND ASH FREE COAL.

10	Volatile.....per cent..	47.6	37.5	21.55
11	Fixed carbon.....do.....	52.4	62.5	78.45

FACTORS INDICATING BURNING QUALITIES.

12	Volatile carbon+available hydrogen.....	6.9	5.1	2.24
13	Product of lines 10 and 12.....	327	191	48.3
14	Line 7÷line 5.....	0.178	0.089	0.049

THE MOST ECONOMICAL AIR SUPPLY FOR A GIVEN FURNACE AND GIVEN FUEL.

For a given furnace and a given fuel there is a certain percentage of excess air which gives the maximum over-all efficiency of a steam-generating apparatus. If the supply of air is increased beyond this percentage the over-all efficiency drops because of the heat lost in heating this extra air. If, on the other hand, the air supply is decreased below this best percentage, the heat losses increase on account of less complete combustion. These are the two principal causes of heat not being available for a boiler and the ones that are affected by air supply. The percentage of excess air giving the lowest sum of these two combined losses varies with the size of the combustion space and the kind of coal. When the combustion space is large smaller air excess is necessary to obtain good combustion than when the combustion space is small. With coals having low percentages of volatile matter less excess air is needed for nearly complete combustion than with coals having high percentages of volatile matter. Figures 35 to 39, inclusive, show that these statements are true. These figures were compiled from figures 24 to 30, inclusive. In figures 35 to 38 the abscissas are the percentages of air in excess of the amount theoretically needed to oxidize completely all carbon to CO_2 and all hydrogen to H_2O . The lower scale gives the corresponding percentages of CO_2 in the products of combustion. The ordinates are the percentages of heat not available for the boiler. The straight inclined line gives the quantity of heat that it takes to raise the temperature of the dry air, supplied for combustion, from atmospheric temperature to the temperature of saturated steam under 150-pound gage pressure. The dotted curves in the lower part of the figure give the undeveloped heat in the gaseous combustible at the end of that part of the combustion space having a ratio to the grate area indicated by the figure at each curve. Thus, for example, the curve designated by 3 represents a furnace having 3 cubic feet of combustion space to every square foot of grate area, and so on. Each of the solid curves in the upper part of the figure give the sum of the quantities of heat shown by the straight line and one of the dotted curves; that is, it shows the quantity of heat not available for the boiler due to both incomplete combustion and the sensible heat in the dry furnace gases. The heat in the moisture of the coal and that formed by the burning of hydrogen does not vary with the air supply and therefore does not affect the shape nor the horizontal position of these curves. Similarly the change in the boiler pressure does not change the slope of the straight line; it merely raises or lowers the line. Inasmuch as it is desired to show the minima in the upper curves, or in other words the percentage of excess air giving

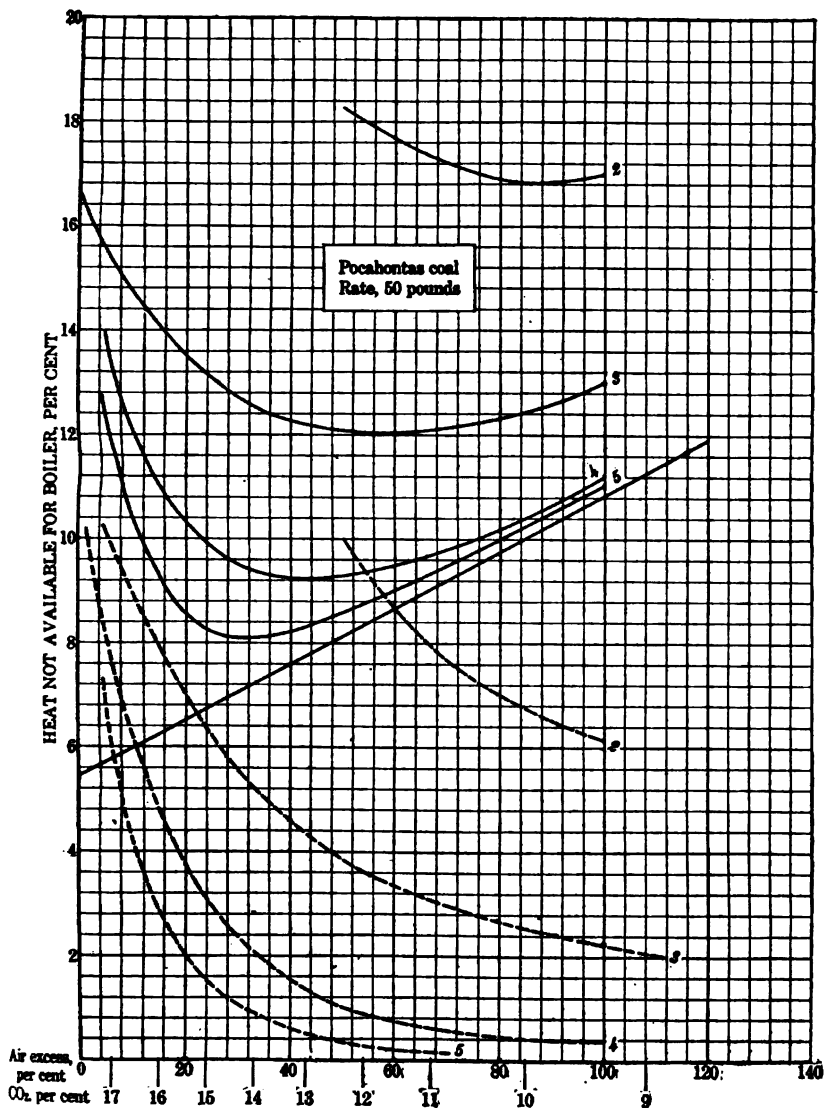


FIGURE 35.—Effect of air supply on the two principal sources of heat loss in a steam-generating apparatus as shown by tests of Pocahontas coal; rate of firing, 50 pounds. The straight inclined line indicates the increase in quantity of heat absorbed in heating the air supply from atmospheric temperature to the temperature of steam in the boiler. The dotted curves show percentage of heat not developed, owing to incomplete combustion, in a furnace having the ratio of combustion space to grate area indicated by the figure near each curve. The solid-line curves represent the sum of the heat losses shown by the straight line and each of the dotted curves, and show what the excess of air should be for the minimum heat losses in a given furnace.

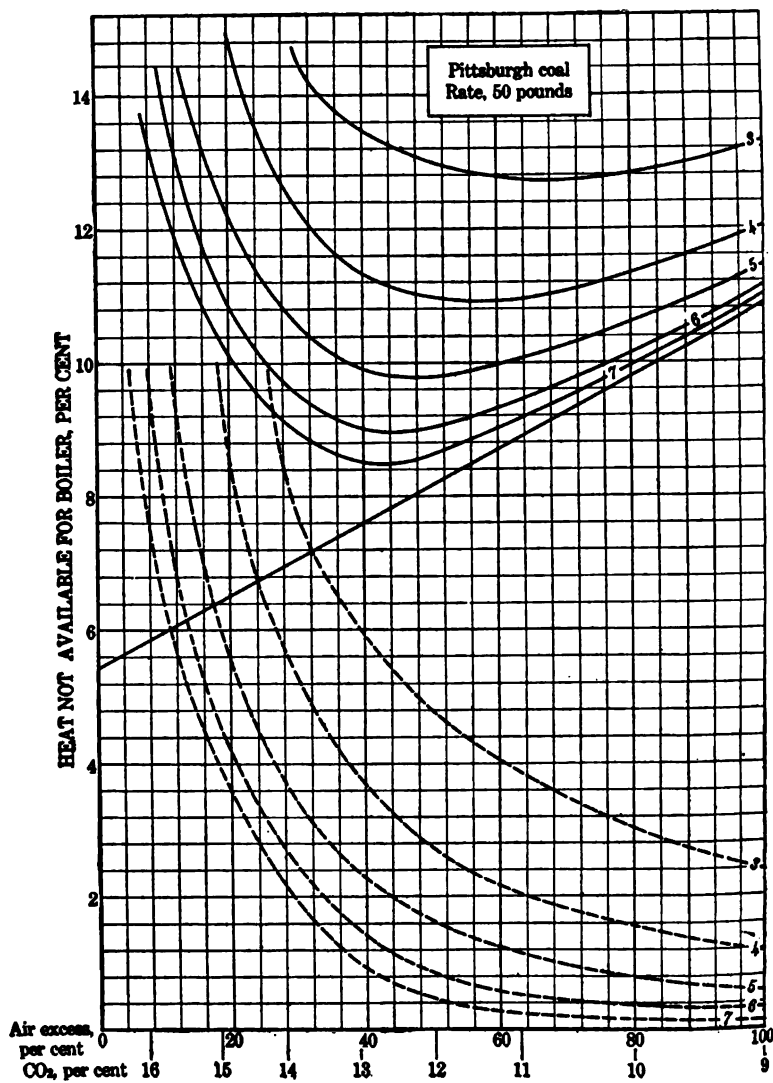


FIGURE 36.—Effect of air supply on the two principal sources of heat loss in a steam-generating apparatus as shown by tests of Pittsburgh coal; rate of firing, 50 pounds. The straight inclined line indicates the increase in quantity of heat absorbed in heating the air supply from atmospheric temperature to the temperature of steam in the boiler. The dotted curves show percentage of heat not developed, owing to incomplete combustion, in a furnace having the ratio of combustion space to grate area indicated by the figure near each curve. The solid-line curves represent the sum of the heat losses shown by the straight line and each of the dotted curves, and show what the excess of air should be for the minimum heat losses in a given furnace.

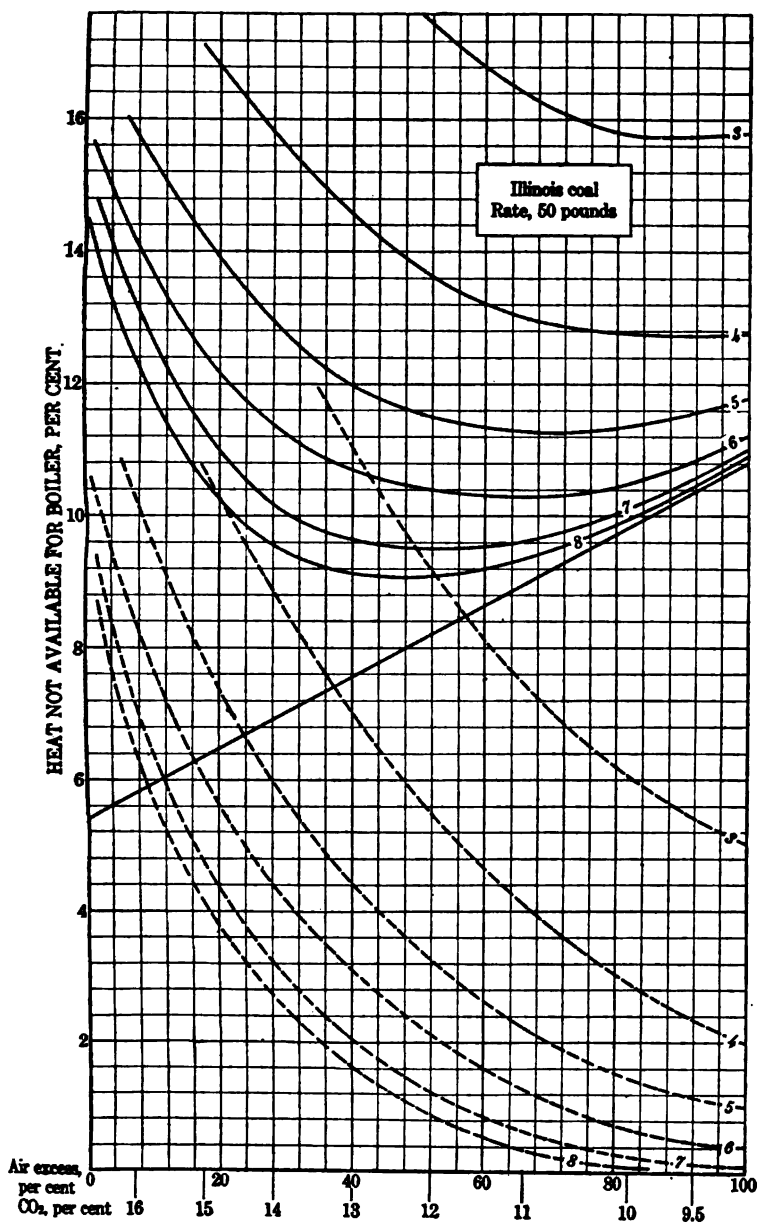


FIGURE 37.—Effect of air supply on the two principal sources of heat loss in a steam-generating apparatus as shown by tests of Illinois coal; rate of firing, 50 pounds. The straight inclined line indicates the increase in quantity of heat absorbed in heating the air supply from atmospheric temperature to the temperature of steam in the boiler. The dotted curves show percentage of heat not developed, owing to incomplete combustion, in a furnace having the ratio of combustion space to grate area indicated by the figure near each curve. The solid-line curves represent the sum of the heat losses shown by the straight line and each of the dotted curves, and show what the excess of air should be for the minimum heat losses in a given furnace.

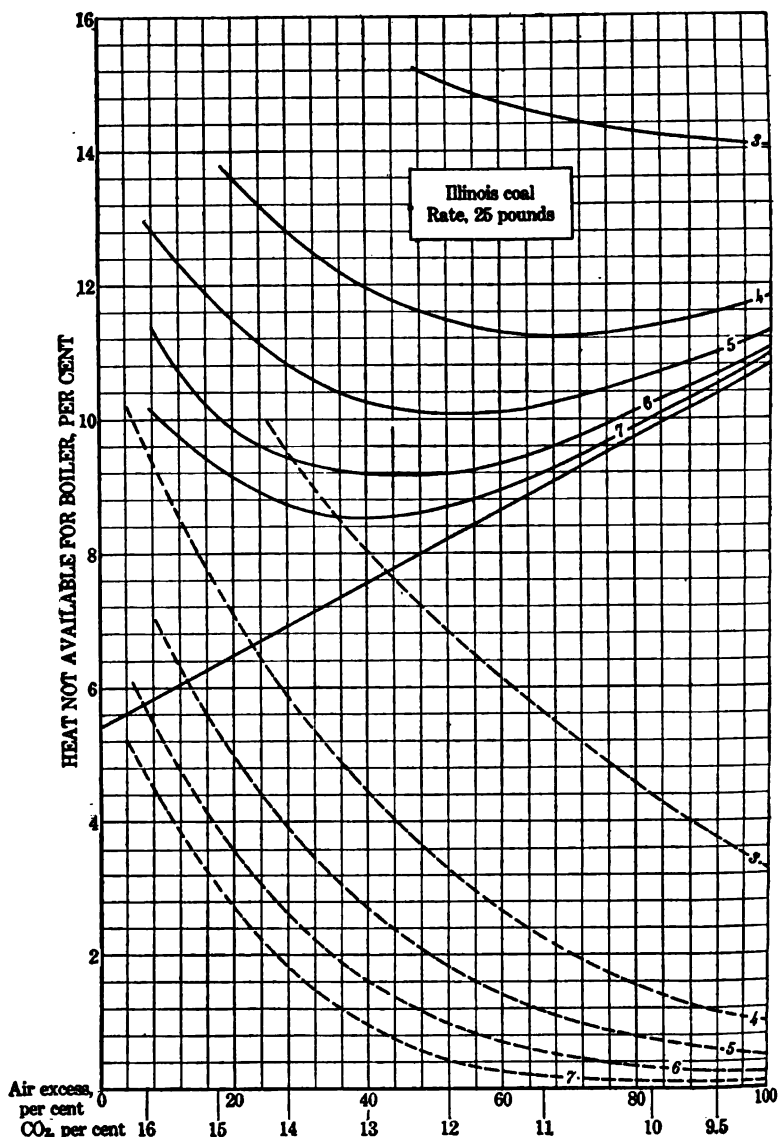


FIGURE 38.—Effect of air supply on the two principal sources of heat loss in a steam-generating apparatus as shown by tests of Illinois coal; rate of firing, 25 pounds. The straight inclined line indicates the increase in quantity of heat absorbed in heating the air supply from atmospheric temperature to the temperature of steam in the boiler. The dotted curves show percentage of heat not developed, owing to incomplete combustion, in a furnace having the ratio of combustion space to grate area indicated by the figure near each curve. The solid-line curves represent the sum of the heat losses shown by the straight line and each of the dotted curves, and show what the excess of air should be for the minimum heat losses in a given furnace.

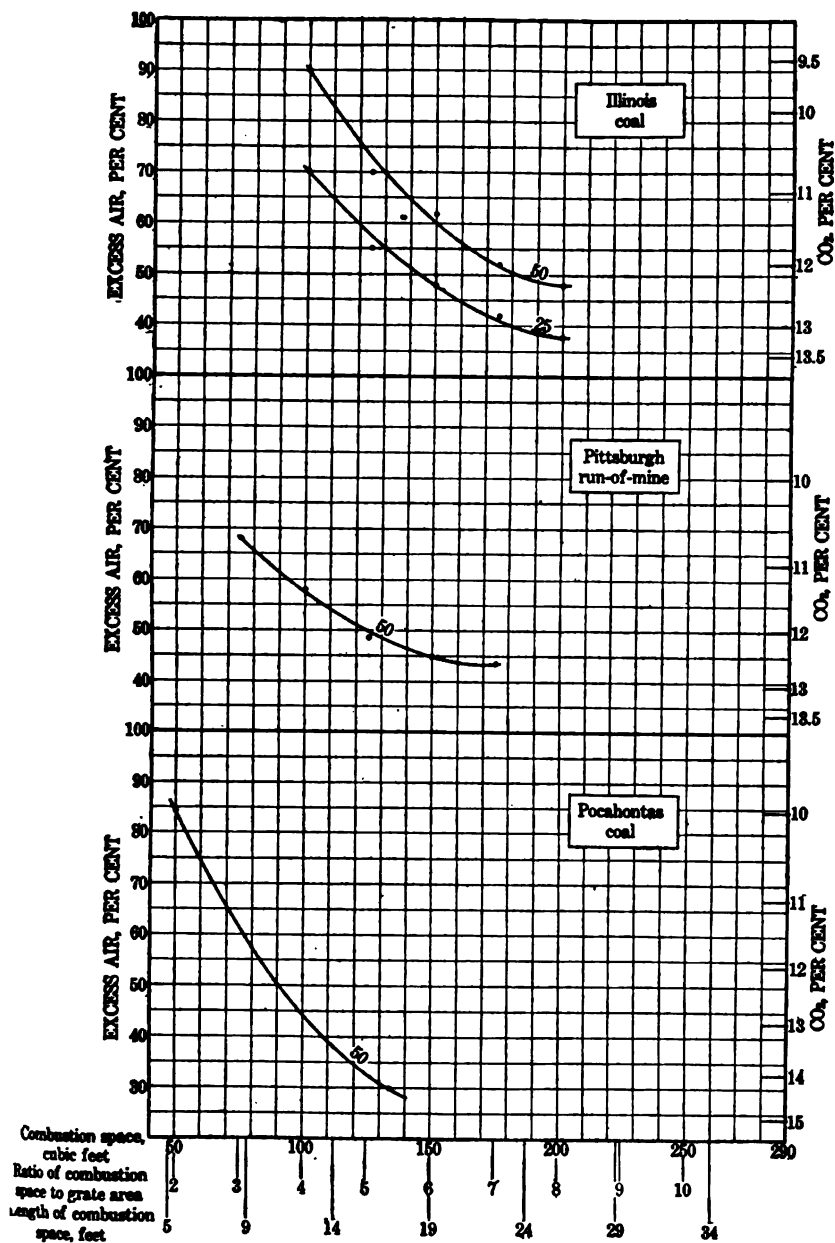


FIGURE 28.—Curves showing percentage of excess air giving the minimum heat losses in burning Pittsburgh, Pocahontas, or Illinois coal in various sizes of combustion space. Proportions of combustion space given by the scales at the bottom. Scale at right indicates percentage of CO₂ in furnace gases for any given excess of air shown by scale at left. Figure near each curve indicates the rate of firing the coal in pounds per square foot of grate per hour.

the least losses, and therefore the best results, the moisture and the steam pressure need not be taken into consideration in compiling such curves. That is, only those losses directly affected by the excess of air are considered here. These losses are expressed in percentage of the calorific value of coal.

The upper curves of figures 35 to 38, inclusive, show that as the size of the combustion space increases the minimum losses are obtained with a lower excess of air and a higher percentage of CO₂ in the furnace gases. The minimum losses in the furnace having a small combustion space are much larger than the minimum losses in the furnace equipped with a large combustion space. However, with the large combustion space the minimum losses extend over a much smaller range of excess air than they would with a small combustion space. This means that with a furnace having a large combustion space more skill is required to keep its performance within the narrow range of minimum losses or maximum efficiency than to operate a small furnace at its best. With the furnace having a small combustion space a variation of 50 to 100 per cent in the excess of air, and even more, makes very little difference in the performance of the furnace. However, the maximum efficiency of the furnace having a large combustion space is so much higher than that of the furnace with a small combustion space that there is little doubt left as to which furnace is preferable.

Figures 35, 36, and 37 were prepared from tests of the three different coals at the same rate of combustion—50 pounds per square foot of grate per hour. Comparison of the curves in the three figures shows that with the same size of combustion space considerably more air is needed for the best results when burning high-volatile coal, such as Illinois, than when using the low-volatile Pocahontas coal. Thus with a furnace having 5 cubic feet of combustion space per square foot of grate area the best results with the Illinois coal were obtained when the excess of air was about 70 per cent, whereas with the Pocahontas coal the best results required only about 30 per cent of excess air. However, this best performance with Illinois coal appears to be about 3 per cent of the total heat in coal lower than the best results with the Pocahontas coal.

To show the effect of the rate of combustion on the excess of air giving the best results under a boiler, figure 38 has been prepared. This figure gives the conditions for the best results when burning Illinois coal in the experimental furnace at the rate of 25 pounds per square foot of grate per hour. Comparison of figure 38 with figure 37 shows that with the lower rate of combustion the minimum heat losses occur with somewhat smaller excess of air and the losses are a little lower.

Figure 39 shows the relation between the size of the combustion space and the percentage of excess air giving the best results under

a steam boiler. The figure consists of three separate diagrams having common abscissas, each diagram giving the relation for one of the coals tested. The diagram for the Illinois coal, given at the top of the figure, has two curves, one for a rate of combustion of 50 pounds, and one for a rate of 25 pounds, of coal per square foot of grate per hour. The other two diagrams show only one curve for a rate of combustion of 50 pounds. The scale on the left gives the percentage of excess air and the scale on the right the corresponding percentage of CO_2 in the furnace gases. The size of the combustion space at the bottom of the chart is given by the three scales. Reading from the top the first scale gives the combustion space in cubic feet, the second scale below gives the number of cubic feet of combustion space for each square foot of grate area, and the third scale is the average length of the path of gas travel through the combustion space.

The most important feature shown by figure 39 is that different furnaces and different fuels require different percentages of excess air to give the best results. This means that there is no one definite percentage of CO_2 in the furnace gases that will give the best results in every furnace burning any coal. Some furnaces and some fuel may give best results with high percentages of CO_2 , whereas others may have to be operated with a comparatively low CO_2 content. Figure 39 may furnish a guide as to the probable best CO_2 content for a given furnace and a given coal.

DISCUSSION OF MISCELLANEOUS DATA.

STRATIFICATION OF GASES.

There seems to be a considerable tendency for furnace gases to flow in stratified streams through the combustion space. The stratification occurs in spite of the whirling of the gases above the fuel bed of the Murphy furnace. The whirling was plainly visible through a peephole in the side wall of the Heine boiler. This whirling motion is caused by the two streams of air coming in through the tuyères and the stream of gases rising from the center of the fuel bed. It seems to continue for some distance beyond the fuel bed. The motion, and the three streams of gases causing it, are shown in figure 7. It would seem that such a motion would cause the gases to form a uniform mixture a short distance beyond the fuel bed. That such is not the case is shown by the variation in the composition of gas samples taken simultaneously at different points of the same cross section, as shown in Table 8, which gives the analyses of all the gas samples taken in test 188, and Table 9, giving the analyses of samples taken in test 195. In each table column 1 gives the section and column 2 the location of the sampler within the section where the gases were collected.

TABLE 8.—Results of analysis of individual gas samples collected in test 188, with Pittsburgh screenings. Rate of combustion 38 pounds.

Section of furnace where sample was taken.	Location of sampler within section.	Percentage, by volume, of—								Tar, grains per cubic foot of gas at 80° F. and 30-inch pressure.	Soot, grains per cubic foot of gas at 80° F. and 30-inch pressure.
		CO ₂	Unsat- urated hydro- carbons.	O ₂	CO	CH ₄	H ₂	N ₂	Total gaseous combust- ible.		
1	2	3	4	5	6	7	8	9	10	11	12
At surface of fuel bed...	L-24	7.5	0.4	0.9	18.8	1.4	10.0	61.0	80.8		
Do.....	L-40	8.7	.0	4.2	9.6	.4	5.1	72.0	15.1		
Do.....	L-56	6.1	.0	14.2	.9	.0	.2	79.6	1.1		
Do.....	M-24	1.3	.0	.4	33.1	.2	3.6	61.4	36.9		
Do.....	M-40	1.8	.0	.0	32.1	.0	2.6	63.5	34.7		
Do.....	M-56	8.6	.0	12.0	.0	.0	.1	79.3	.1		
Do.....	R-24	10.5	.0	6.2	7.6	.3	3.1	75.3	11.0		
Do.....	R-40	9.2	.3	6.5	5.2	.9	5.6	72.3	12.0		
Do.....	R-56	11.6	.3	8.5	.1	.0	.1	79.7	.2		
Average.....		7.1	.1	5.5	11.9	.4	3.4	71.6	15.8	0.107	0.080
11 inches above fuel bed.	L-24	10.6	.0	5.4	3.1	.0	2.6	78.8	5.7		
Do.....	L-40	8.5	.0	1.2	11.3	1.9	9.9	67.2	23.1		
Do.....	L-56	14.0	.0	.8	4.0	.1	1.9	79.2	6.0		
Do.....	M-24	2.8	.0	.2	29.5	.2	4.4	62.9	34.1		
Do.....	M-40	10.2	.0	.3	16.0	.0	3.0	70.5	19.0		
Do.....	M-56	10.0	.0	6.4	5.9	.1	1.6	76.0	7.6		
Do.....	R-24	8.8	.0	8.4	1.0	.0	1.6	80.2	2.6		
Do.....	R-40	9.8	.0	6.8	1.4	.2	1.2	80.6	2.8		
Do.....	R-56	11.5	.0	6.6	.8	.0	.2	80.9	1.0		
Average.....		9.6	.0	4.0	8.1	.2	2.9	75.1	11.3	0.013	0.284
22 inches from fuel bed.	M-24	12.1	.0	1.7	10.2	.1	1.9	74.0	12.2		
Do.....	M-40	12.1	.0	2.2	8.1	.1	1.5	76.0	9.7		
Do.....	M-56	12.8	.0	3.2	5.6	.0	1.3	77.1	6.9		
Average.....		12.3	.0	2.4	8.0	.1	1.6	75.6	9.6		
Section A.....	L-5	15.6	.0	2.0	1.8	.0	.4	80.7	1.7		
Do.....	L-16	12.1	.0	.2	8.4	.1	3.4	75.8	11.9		
Do.....	L-21	.8	.1	.0	22.7	.1	4.5	61.8	37.3		
Do.....	M-5	15.1	.0	3.8	1.0	.0	.5	79.5	1.5		
Do.....	M-16	14.4	.0	2.3	5.7	.0	1.4	76.2	7.1		
Do.....	M-21	.6	.1	.0	33.4	.1	4.6	61.2	38.1		
Do.....	R-5	13.3	.0	5.9	.3	.0	.1	80.4	.4		
Do.....	R-16	14.3	.0	2.0	2.6	.0	1.2	79.9	3.8		
Do.....	R-21	.1	.1	.1	33.7	.0	4.2	61.8	37.9		
Average.....		14.2	.0	2.7	8.2	.0	1.2	78.7	4.4	0.014	0.160
Section C.....	L-5	14.6		4.6	.1	.0	.1	80.6	.2		
Do.....	L-16	15.3		3.1	.4	.0	.2	81.0	.6		
Do.....	L-27	14.3		4.3	.1	.0	.2	81.1	.3		
Do.....	M-5	14.9		3.5	.4	.0	.1	81.1	.5		
Do.....	M-16	14.4		3.7	.3	.0	.1	81.5	.4		
Do.....	M-27	14.0		3.9	.2	.0	.0	81.9	.2		
Do.....	R-5	12.1		7.0	.1	.0	.0	80.8	.1		
Do.....	R-16	15.0		3.2	.3	.0	.1	81.4	.4		
Do.....	R-27	13.9		4.7	.3	.0	.1	81.0	.4		
Average.....		14.3		4.2	.2	.0	.1	81.2	.3		
Section E.....	L-5	13.8		5.0	.0	.0	.0	81.2	.0		
Do.....	L-16	14.8		3.8	.2	.0	.0	81.2	.2		
Do.....	L-27	14.9		3.7	.2	.0	.0	81.2	.2		
Do.....	M-5	14.0		4.8	.1	.0	.0	81.1	.1		
Do.....	M-16	14.7		3.7	.4	.0	.0	81.2	.4		
Do.....	M-27	14.3		4.5	.4	.0	.0	80.8	.4		
Do.....	R-5	13.7		5.2	.1	.0	.0	81.0	.1		
Do.....	R-16	15.0		3.4	.3	.0	.0	81.3	.3		
Do.....	R-27	11.7		7.3	.3	.0	.0	80.7	.3		
Average.....		14.1		4.6	.2	.0	.0	81.1	.2		

TABLE 8.—Results of analysis of individual gas samples collected in test 188, with Pittsburgh screenings. Rate of combustion 58 pounds—Continued.

Section of furnace where sample was taken.	Location of samples within section.	Percentage, by volume, of—								Tar, grams per cubic foot of gas at 60° F. and 30-inch pressure.	Soot, grams per cubic foot of gas at 60° F. and 30-inch pressure.
		CO ₂	Unsaturated hydrocarbons.	O ₂	CO	CH ₄	H ₂	N ₂	Total gaseous combustible.		
1	2	3	4	5	6	7	8	9	10	11	12
Section G.....	L-5	14.8		3.5	.3	.0	.0	81.4	.3		
Do.....	L-16	15.0		3.5	.2	.0	.0	81.3	.2		
Do.....	L-27	14.9		3.6	.3	.0	.0	81.2	.3		
Do.....	M-5	14.8		3.6	.3	.0	.0	81.3	.3		
Do.....	M-16	14.1		4.3	.2	.0	.0	81.4	.2		
Do.....	M-27	14.9		3.6	.3	.0	.0	81.2	.3		
Do.....	R-5	13.1		5.4	.1	.0	.0	81.4	.1		
Do.....	R-16	15.0		3.4	.3	.0	.0	81.3	.3		
Do.....	R-27	14.8		3.5	.4	.0	.0	81.3	.4		
Average.....		14.6		3.8	.3	.0	.0	81.3	.3		

TABLE 9.—Results of analysis of individual gas samples collected in test 195, with Pittsburgh screenings. Rate of combustion 24 pounds.

Section of furnace where sample was taken.	Location of samples within section.	Percentage, by volume, of—								Tar, grams per cubic foot of gas at 60° F. and 30-inch pressure.	Soot, grams per cubic foot of gas at 60° F. and 30-inch pressure.
		CO ₂	Unsaturated hydrocarbons	O ₂	CO	CH ₄	H ₂	N ₂	Total gaseous combustible.		
1	2	3	4	5	6	7	8	9	10	11	12
At surface of fuel bed...	L-24	5.1	0.4	0.0	22.2	1.3	11.8	59.2	35.7	0.190	0.259
Do.....	L-40	13.6	.0	.5	5.0	.0	2.2	78.6	7.3	.022	.194
Do.....	L-56	7.4	.0	.2	16.9	.5	9.5	65.5	26.9	.038	.344
Do.....	M-24	2.7	.0	.2	31.5	.4	6.4	58.8	38.5		
Do.....	M-40	7.8	.0	2.8	16.6	.4	2.2	70.2	19.2		
Do.....	M-56	12.9	.0	5.0	3.9	.1	.7	77.4	4.7		
Do.....	R-24	9.2	.1	.4	16.5	.9	4.6	68.3	22.1	.059	.068
Do.....	R-40	10.4	.1	1.3	10.7	1.0	6.1	70.4	17.9	.006	.032
Do.....	R-56	10.6	.2	.6	8.1	1.1	7.4	72.0	16.8	.004	.182
Average.....		8.9	.1	1.2	14.6	.6	5.7	68.9	21.0	.054	.180
11 inches above fuel bed..	L-24	13.2	.0	1.7	4.4	.4	1.7	78.6	6.5	.009	.197
Do.....	L-40	7.8	.0	.2	17.9	.5	6.5	67.1	24.9	.009	.152
Do.....	L-56	14.6	.0	.6	2.8	.0	1.6	80.4	4.4	.028	.284
Do.....	M-24	6.3	.0	.3	23.8	.1	4.2	65.3	28.1		
Do.....	M-40	10.6	.0	1.9	13.4	.3	1.9	71.9	15.6		
Do.....	M-56	12.6	.0	1.0	10.2	.2	1.7	74.3	12.1		
Do.....	R-24	10.3	.0	5.2	1.9	.6	1.1	80.9	3.6	.003	.063
Do.....	R-40	8.9	.0	8.8	.0	.0	.1	82.2	.1	.006	.120
Do.....	R-56	12.7	.0	3.1	3.0	.2	1.1	79.9	4.3	.004	.178
Average.....		10.8	.0	2.5	8.6	.3	2.2	75.6	11.1	.010	.169
22 inches from fuel bed..	M-24	11.7	.0	7.5	1.2	.0	.2	79.4	1.4		
Do.....	M-40	12.5	.1	6.9	.4	.0	.2	79.9	.7		
Do.....	M-56	10.8	.0	8.6	.2	.0	.0	80.4	.2		
Average.....		11.7	.0	7.7	.6	.0	.1	79.9	.9		

TABLE 9.—*Results of analysis of individual gas samples collected in test 195, with Pittsburgh screenings. Rate of combustion 24 pounds—Continued.*

Section of furnace where sample was taken.	Location of sampler within section.	Percentage, by volume, of—								Tar, grams per cubic foot of gas at 60° F. and 30-inch pressure.	Soot, grams per cubic foot of gas at 60° F. and 30-inch pressure.
		CO ₂ .	Unsat- urated hydro- carbons.	O ₂ .	CO.	CH ₄ .	H ₂ .	N ₂ .	Total gaseous combust- ible.		
1	2	3	4	5	6	7	8	9	10	11	12
Section A.....	L-10	15.1	.0	2.4	.7	.0	.5	81.3	1.2	.002	.075
Do.....	L-21	11.5	.0	5.3	2.7	.1	.8	79.6	3.6	.001	.040
Do.....	M-10	13.7	.0	.5	8.4	.2	1.5	75.7	10.1	.002	.072
Do.....	M-21	16.1	.0	1.7	2.7	.1	.7	78.7	3.5	.001	.068
Do.....	R-10	14.2	.0	1.2	4.9	.3	.8	78.6	6.0	.003	.183
Do.....	R-21	10.6	.0	1.1	8.4	.5	3.9	75.5	12.8	.013	.327
Average.....		13.5	.0	2.0	4.6	.2	1.4	78.3	6.2	.004	.128
Section C.....	L- 5	12.3		6.2	.4	.0	.0	81.1	.4		
Do.....	L-16	16.3		1.3	.8	.0	.2	81.4	1.0		
Do.....	L-27	15.5		2.5	.8	.0	.2	81.0	1.0		
Do.....	M- 5	16.1		2.1	1.0	.0	.5	80.3	1.5		
Do.....	M-16	15.8		1.5	1.5	.1	.5	80.6	2.1		
Do.....	M-27	14.6		3.0	1.0	.0	.6	80.8	1.6		
Do.....	R- 5	14.7		3.9	.5	.0	.1	80.8	.6		
Do.....	R-16	15.8		2.2	.7	.0	.2	81.1	.9		
Do.....	R-27	15.3		2.9	.4	.0	.3	81.1	.7		
Average.....		15.2		2.8	.8	.0	.3	80.9	1.1		
Section E.....	L- 5	13.1		5.4	.2	.0	.1	81.2	.3		
Do.....	L-16	16.2		1.4	.8	.0	.4	81.2	1.2		
Do.....	L-27	16.2		1.7	.6	.0	.1	81.4	.7		
Do.....	M- 5	15.8		2.9	.0	.0	.0	81.3	.0		
Do.....	M-16	16.0		1.6	.8	.0	.3	81.3	1.1		
Do.....	M-27	16.2		1.6	.8	.0	.3	81.1	1.1		
Do.....	R- 5	16.4		1.4	1.3	.0	.3	80.6	1.6		
Do.....	R-16	16.1		1.1	1.8	.0	.7	80.3	2.5		
Do.....	R-27	16.1		1.2	1.8	.0	.6	80.3	2.4		
Average.....		15.8		2.0	.9	.0	.3	81.0	1.2		
Section G.....	L- 5	16.1		1.6	1.1	.0	.3	80.9	1.4		
Do.....	L-16	15.1									
Do.....	L-27	16.0		1.7	.9	.0	.2	81.2	1.1		
Do.....	M- 5	15.8		1.7	1.1	.0	.3	81.1	1.4		
Do.....	M-16	16.1		1.3	1.3	.0	.5	80.8	1.8		
Do.....	M-27	16.4		1.3	1.1	.0	.4	80.8	1.5		
Do.....	R-16	16.2		1.0	1.5	.0	.6	80.7	2.1		
Do.....	R-27	15.7		1.5	1.5	.0	.5	80.8	2.0		
Average.....		15.9		1.4	1.2	.0	.4	81.1	1.6		

Sections A to G are vertical; the three sections preceding section A in the tables are nearly horizontal planes parallel to and directly over the surface of the fuel bed. In Table 8 the first group of nine gas samples was taken just at the surface of the fuel bed; the second group was taken 11 inches above the fuel bed; and the third group of three samples 22 inches above the fuel bed. In column 2 the letters L, M, and R stand for left, middle, and right sampler, respectively. The figure following each letter indicates the distance of the intake of the sampler from the inside of the front wall. Thus, L-40 means that the sample was collected with the sampler on the left, 40 inches from the inside of the front wall. The position of the left, middle, and

right samplers is shown in figure 4 (p. 15). Only three samples were collected at the section 22 inches above the fuel bed.

In the series of tests shown in Table 9 only six samples were collected at section A, because the samplers were constructed for collecting tar and soot samples and for that reason each sampler contained only two gas sampling tubes. Beyond section A every alternate section was omitted on this series. The position of the samplers at section A is shown in figure 5 (p. 16). The position of the sampler in the furnace and the distance of the intake of the sampler from the arch is designated similarly to Table 8.

All the sections are nearly at right angles to the path of the gases; above the fuel bed where the gases flow nearly vertically the sections are horizontal; in the combustion chamber where the gases flow horizontally the sections are vertical.

Table 8 shows that even beyond section A in the combustion chamber the gases tend to flow in stratified streams, apparently without any regard to their specific weight. In general, near the roof of the combustion chamber the gases contain high percentages of oxygen and very little combustible; near the bottom they contain high percentages of combustible and very little free oxygen. This rather regular distribution of oxygen at the top and combustible gases at the bottom would seem to indicate that there is a predominance of horizontal layers in the stream of furnace gases. The vertical layers are much more irregular, showing that there is some change of gas from side to side.

This predominance of horizontal layers in the furnace gases as they flow through the combustion chamber is perhaps shown more markedly in Table 10, which gives the excess of air, calculated as shown on page 28, for each sample of gas. The designation of the position of the samplers is similar to that in Tables 3, 8, and 9. In the tests of Table 10 no samples were taken before section A and no tars and soot samples were collected. Nine gas samples were collected at section A.

All the samples collected 5 to 7 inches below the arch show, as a rule, considerable excess of air, whereas the samples collected near the bottom show small excess or even deficient air supply. This difference in the excess of air becomes smaller as the gases flow through the combustion space, until at section H the mixture of gases becomes fairly homogenous.

Tables 4 and 10 demonstrate that it is much more difficult to obtain the average gas sample than it is to analyze it. This is a similar conclusion to the one arrived at from tests described in Bulletin 97.^a

^a Kreisinger, Henry, and Ovits, F. K., Sampling and analysis of flue gases; Bull. 97, Bureau of Mines, 1915, 70 pp.

TABLE 10.—Results showing percentages of excess air calculated for individual simultaneous gas samples, showing stratification of gases.

[Tests with Illinois coal.]

Section of furnace chamber where sample was taken.	Position of sample within section.	Percentage of excess air in samples represented by test No.—																
		165	166	167	168	169	170	171	172	173	174	175	176	177	178	179	180	181
Section A.	L-5.....	98.2	236.5	301.3	328.2	103.1	312.5	260.7	119.2	683.1	301.3	95.8	300.3	70.0	391.7			
	L-16.....	0.9	6.3	69.4	109.7	69.4	17.3	17.4	40.5	58.0	30.1	50.4	21.6	0.0	68.9			414.0
	L-27.....	28.0	60.9	43.3	17.3	46.7	33.0	53.2	36.4	51.8	43.1	20.5	55.6	42.0	46.4			10.7
	M-5.....	38.9	213.7	285.2	192.0	76.4	302.8	235.7	52.2	621.4	186.5	38.3	178.5	8.3	290.0			55.9
	M-16.....	11.0	81.0	148.9	79.2	14.2	146.2	45.5	32.2	23.7	5.4	7.2	92.9	10.7	17.9			339.9
	M-27.....	47.2	49.1	26.7	26.7	37.5	16.3	2.2	27.3	57.1	50.0	21.6	5.5	8.5	6.9			94.4
	R-5.....	493.5	443.5	447.9	568.8	597.3	543.3	543.3	543.3	543.3	543.3	197.6	354.8	63.7	231.0			41.3
	R-16.....	60.8	27.1	60.8	5.9	100.1	245.4	100.1	100.1	100.1	100.1	7.2	9.0	22.3	23.0			321.5
Section B.	R-27.....	1.7	19.2	51.0	12.5	54.7	43.0	59.3	11.8	41.1	45.4	45.8	56.8	41.6	45.1			18.3
	L-5.....	47.8	120.6	167.5	149.2	102.1	207.2	197.1	59.4	270.8	196.8	118.9	167.2	60.2	207.2	293.9	107.8	229.5
	L-16.....	33.0	20.6	81.5	65.4	58.8	52.7	7.9	41.6	42.3	38.0	37.6	27.5	21.0	45.4	40.3	0.5	20.7
	L-27.....	11.3	13.0	59.2	18.2	10.6	21.6	16.9	22.8	9.6		9.6	5.8	7.1	1.4	1.2	25.1	15.1
	M-5.....	40.6	146.8	197.5	207.6	87.6	235.7	179.2	30.4	293.5	163.7	36.4	181.8	18.6	196.8	293.5	116.9	196.8
	M-16.....	21.5	46.4	100.2	73.3	33.6	100.2	65.8	5.0	132.7	77.1	13.8	96.4	5.9	84.9	99.7	16.6	89.7
	M-27.....	14.7	12.7	39.6	23.6	8.3	55.5	3.9	14.4	28.5	11.3	14.1	0.6	21.4	6.6	1.6	20.9	12.6
	R-5.....	85.6	132.3	192.0	159.6	153.3	228.7	276.7	134.8	167.5	178.7	72.1	78.8	46.1	167.5	188.4	167.5	179.5
Section C.	R-16.....	13.8	35.3	54.2	47.4	51.3	68.7	52.9	53.1	69.3	26.8	41.5	84.0	10.5	24.8	27.5	4.4	41.3
	R-27.....		5.1	16.5	0.9	7.2	28.0	1.6	9.2	15.3	7.3	6.8	9.2	0.5	11.9	9.0	29.3	28.0
	L-5.....	89.6	124.0	175.6	175.6	85.6	170.9	155.3	63.9	272.1	212.9	71.5	152.7	61.6	183.0	198.4	73.1	202.5
	L-16.....	26.7	89.5	63.2	63.2	47.3	65.4	16.2	31.2	37.3	41.9	21.4	14.3	20.4	31.2	50.3	16.3	16.3
	L-27.....	9.0	30.1	43.6	50.5	31.9	43.6	1.7	17.8	18.9	1.2	1.9	10.0	15.3	8.0	29.4	16.7	16.3
	M-5.....	24.9	81.5	121.6	121.6	71.6	149.2	111.5	26.5	163.0	130.9	52.0	139.4	16.6	190.2	202.5	56.6	197.5
	M-16.....	67.5	84.6	74.9	74.9	17.6	79.0	41.9	1.5	98.5	36.1	9.1	53.8	7.4	46.7	64.9	9.0	33.9
	M-27.....	26.0	13.1	52.7	24.5	2.2	52.2	14.7	4.6	48.3	8.1	6.3	21.1	16.0	3.5	12.0	19.1	9.2
Section D.	R-5.....	133.0	114.6	100.2	114.6	100.2	142.9	152.1	34.4	114.0	136.9	82.2	92.7	11.5	113.8	175.0	61.2	
	R-16.....	72.5	61.8	61.8	61.8	45.1	84.5	63.5	24.4	88.3	40.3	33.0	21.6	9.6	34.6	51.9	19.3	58.8
	R-27.....	14.1	15.6	43.1	21.8	14.6	54.2	27.0	21.1	50.4	7.9	19.5	40.1	2.1	10.9	16.2	11.3	4.3
	L-5.....	46.1	111.5	111.5	95.6	59.0	107.5	81.9	42.3	106.2	52.2	58.8	81.2	27.3	108.6	145.9	43.5	87.6
	L-16.....	11.4	35.4	89.7	64.3	34.1	82.8	36.8	19.8	71.8	62.2	11.6	41.6	26.0	52.5	56.6	4.4	33.6
	L-27.....	4.4	32.5	93.1	46.3	56.5	70.4	21.1	34.1	47.4	3.5	2.6	16.5	3.3	27.4	28.7	15.1	1.1
	M-5.....	47.3	54.8	129.4	129.4	84.2	155.6	96.6	57.4	145.2	138.3	42.2	189.3	39.7	145.5	166.5	44.5	109.1
	M-16.....	7.1	42.1	126.6	70.2	81.7	111.8	46.3	8.9	96.1	20.7	19.2	62.8	0.0	12.6	38.5	66.2	37.1
Section E.	M-27.....	13.5	46.5	87.3	43.6	15.9	68.9	28.0	9.2	63.6	4.3	5.6	35.6	0.9	12.6	26.3	110.8	8.1
	R-5.....	64.7	86.6	135.1	124.6	99.9	136.0	121.3	55.7	127.1	108.1	54.7	224.5	20.6	111.8	138.2	60.3	183.5
	R-16.....	14.6	43.5	84.0	60.4	46.3	80.4	67.7	33.9	92.5	48.5	33.9	110.2	4.7	52.0	70.4	23.9	72.9
	R-27.....	8.1	18.0	66.5	33.0	27.1	57.4	29.7	26.8	61.8	6.5	18.9	47.4	2.7	17.2	20.0	10.3	9.2

ACCURACY OF AVERAGE COMPOSITION OF GAS SAMPLES.

The average composition of the furnace gases at any section was obtained by giving all the samples taken at one section equal weight. The individual samplers covered approximately equal areas of the cross section; and if the velocity of the gases were uniform all over the section, the volumes of gases sampled by each sampler would be the same, and an arithmetical average would give the true composition of the furnace gases at the section. However, it is certain that such was not the case. It is a known fact that when gases flow through a tube of either square or circular cross section the velocity is much higher at the center than it is near the walls. Consequently one may safely assume that in the combustion chamber the gases moved faster through the center than along the walls. Therefore the central sampler was taking gas from larger volumes than those near the walls. To obtain the correct average composition of gases, the central sampler should be given greater weight. However, obtaining any reliable figures of the respective velocities is so difficult that no attempt was made to correct for the difference in velocity. Usually eight samples were taken within 7 inches of the walls and only one in the center. Even by giving the central sample double weight the average would be changed very little.

The above reasoning and conclusion holds true for the sections beyond A, but for section A itself the conditions are somewhat different. On account of the V shape of the grate the fuel accumulates in the center of the grate to a thickness of 12 inches or more, particularly in the rear. The result is that the bottom three samplers at section A are below the level of the fuel bed. Inasmuch as section A is only about a foot from the rear end of the grate, these three bottom samplers are in a space behind the fuel bed where the gases are eddying and have a very low forward velocity as compared to the gases near the roof of the furnace. Furthermore this comparatively quiet place back of the fuel bed contains practically gases as they leave the fuel bed without any air mixed with them, all the additional air being introduced higher up above the fuel bed. The air can not in the short length of travel reach the gases in this quiet space. As a result of these conditions the bottom three samples contain much more combustible gas than the samples taken higher up. By giving the bottom samples the same weight as those higher up the average is too high in combustible gases. There is no reliable way to correct this error. As a result of it the excess of air computed from the average analysis of section A will be too low and may make it appear that air has been introduced between sections A and B.

In the series of tests of Pittsburgh screenings only six gas samples were taken at section A; three of these were about 11 inches and three 22 inches below the arch. It is believed that these six samples

gave a better average than the nine samples taken in the tests of the other three series.

RELATION BETWEEN PROPORTION OF CO₂ IN THE FURNACE GASES AND THE EXCESS OF AIR.

With each coal there is a definite relation between the percentage of CO₂ in the furnace gases and the excess of air supplied to the furnace, provided that the amount of combustible in the gases is negligible. Figure 40 shows such a relation for the four coals tested. The points for the curves are obtained by plotting the values of column 7 of Table 3 as ordinates and those of column 19 as the abscissas. All the values are based on samples of gases taken at sections G or H. All four groups of points fall closely along a smooth curve, showing that the relation is definite enough to be of practical use.

With the Pocahontas coal it is possible with 5 per cent excess of air to get a maximum of about 17 per cent of CO₂ in the furnace gases. With the Pittsburgh and Illinois coals it is possible to obtain only about 15½ per cent as the maximum with about 10 per cent excess of air. When the air is reduced below these minima, the percentage of CO₂ drops and combustible gases appear in the flue gases. This is shown by the one point in the group of the Pocahontas coal and the three points of the group of Pittsburgh screenings at the extreme left of the figure. It should be remembered that these results were obtained with a combustion space about 40 feet long. With the combustion space shorter such high percentages of CO₂ could not be obtained. With very small combustion space the maximum CO₂ content obtainable may not be over 12 per cent. That the advantages of large combustion space is appreciated by modern furnace designers is shown by the fact that at one new plant, at Connors Creek, Detroit, Mich., the combustion chamber is 33 feet high.

These facts and the discussion presented in connection with figures 35 to 39, inclusive, indicate that in the setting of horizontal tubular boilers the practice of making the gases hug the shell to increase the over-all efficiency is questionable. Of course, by squeezing the gases against the shell the heat transmission from the gases to the shell is increased; but by doing so the furnace is robbed of many cubic feet of combustion space. Rapid cooling of the gases before the combustion is complete is not desired, because such cooling retards combustion and if carried too far may even prevent it.

RELATION BETWEEN THE PERCENTAGE OF CO AND THE TOTAL COMBUSTIBLE GASES.

The relation between the percentage of CO in the furnace gases and the percentage of total combustible gases is definite enough to be of value in estimating the total combustible from the determination of

CO. Such relation is given in figure 41, each curve applying to one of the four coals tested. The relation varies slightly with the kind of

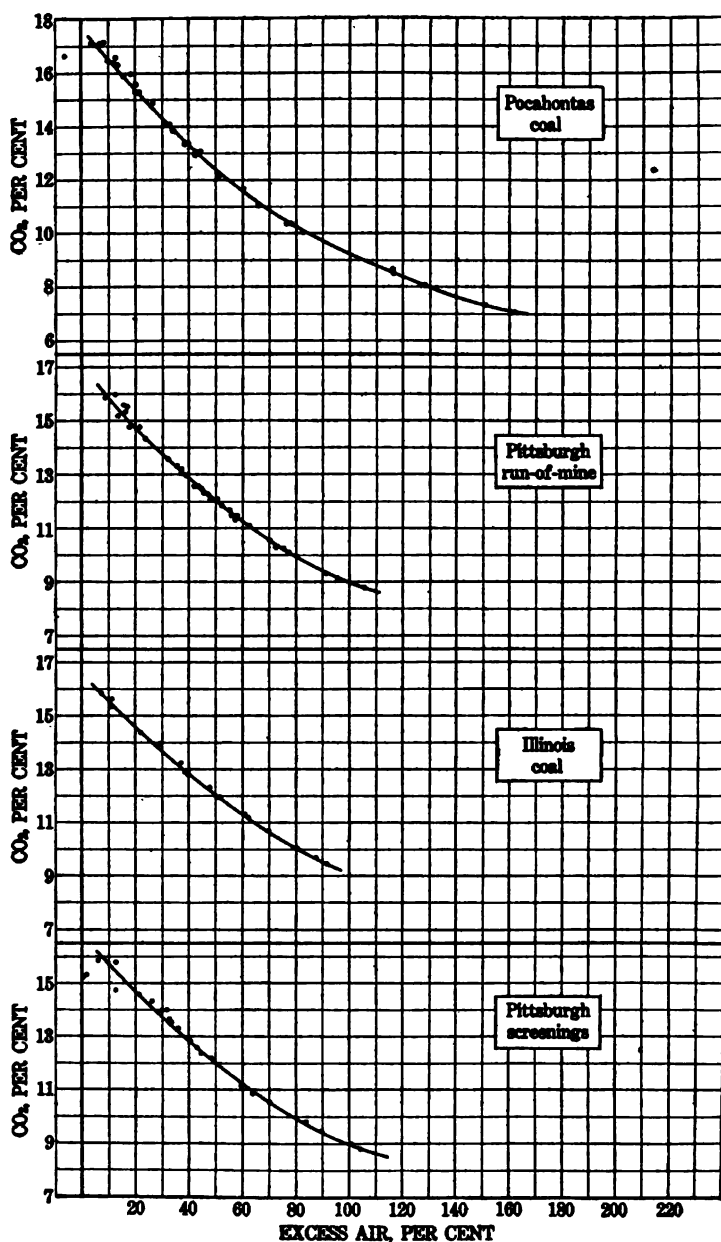


FIGURE 40.—Relation between the percentage of CO_2 in the gases and the percentage of excess of air used in combustion.

coal. Roughly speaking the CO constitutes 80 per cent of the total combustible gases; that is, when the CO content is 2 per cent the con-

tent of total combustible gases is 2.5 per cent, and when the CO content is 0.8 per cent the content of total combustible is 1.0 per cent.

The combustible gases other than CO consist largely of hydrogen; CH_4 , and the unsaturated hydrocarbons constitute a very small part of the gases.

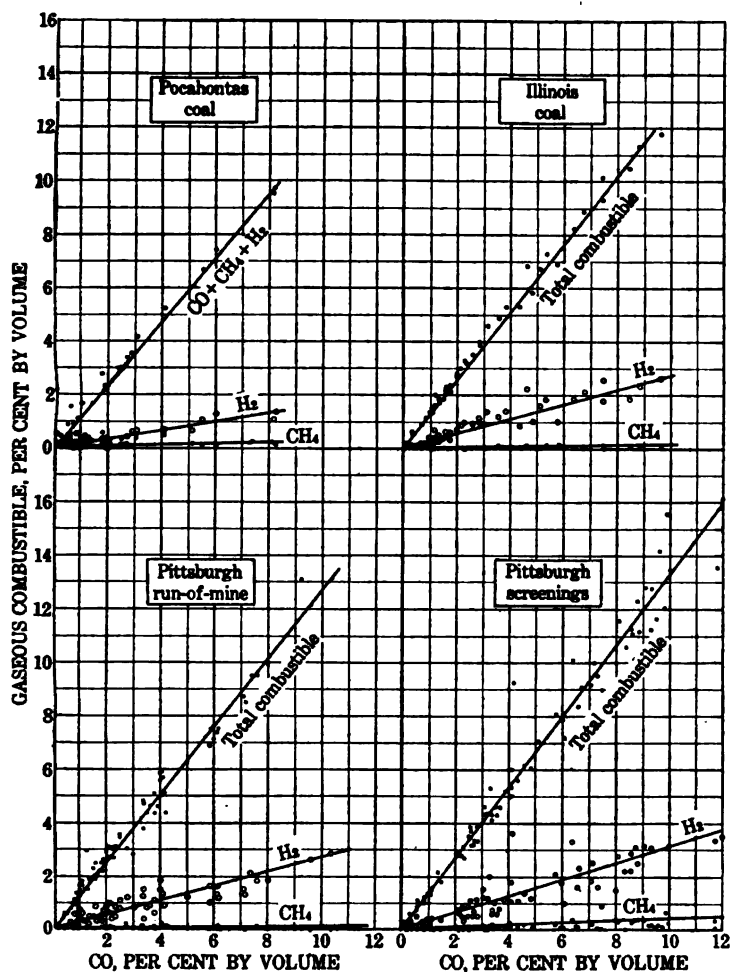


FIGURE 41.—Relation between percentages of CO and H_2 , CH_4 , and total gaseous combustible in furnace gases. Each of the group of curves is for one of the four coals tested.

The small percentage of CH_4 shown in the figure is in accordance with the nature of the gas. It has been shown by various investigators that in absence of air methane decomposes readily into hydrogen and carbon (soot) at high temperatures such as exist in furnaces.

As indicated in Table 11 (p. 113), the percentage of methane that is stable at the high furnace temperature is very low. It is doubtful if methane under ordinary furnace conditions ever reaches equilibrium. The methane content of the gases leaving the fuel bed is

probably much higher than that corresponding to equilibrium conditions, and its amount is rapidly reduced both by combustion and decomposition.

RELATION BETWEEN CO CONTENT IN GASES AND AMOUNT OF UNDEVELOPED HEAT.

Inasmuch as there is a close relation between the proportion of CO and the total gaseous combustible in the furnace gases, there is simi-

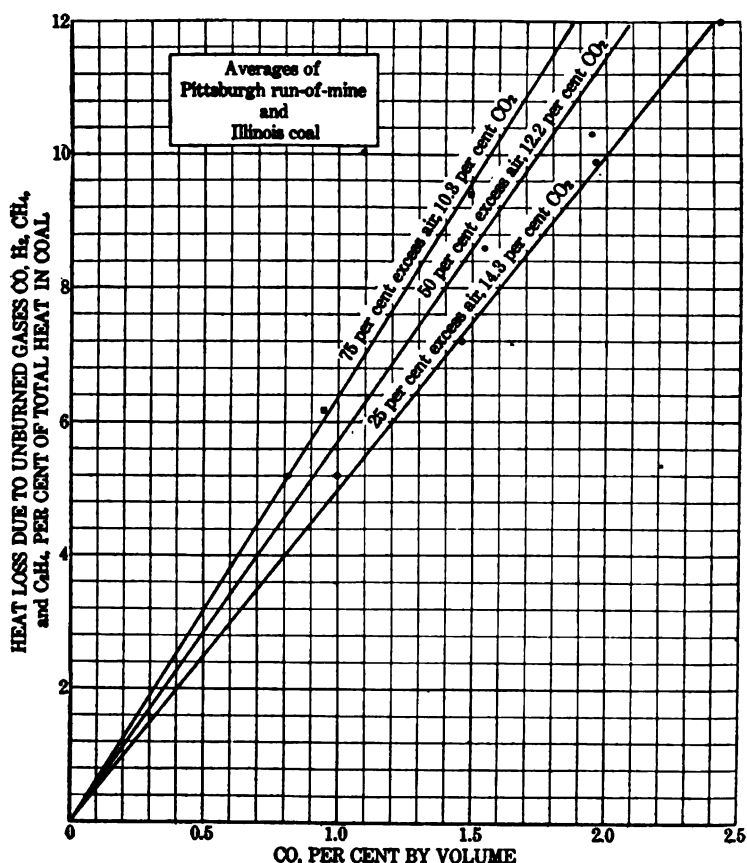


FIGURE 42.—Relation between CO content in furnace gases and heat loss due to the escape of all unburned gases.

larly a close relation between the CO content and the undeveloped heat of the gaseous combustible. This relation is definite enough so that the percentage of CO can be used to determine the heat losses due to the escape of all the unburned combustible gases from the furnace. Figure 42 shows this relation graphically. The ordinates of the figure are the undeveloped heat of the combustible gases expressed in percentage of the total heat in the coal, and the abscissas are the percentage of CO in the furnace gases. Each of the three curves show

this relation for one constant air excess, or percentage of CO_2 in the furnace gases.

Thus, supposing that the gases contain 0.6 per cent of CO and 14.3 per cent of CO_2 , or that the air excess is 25 per cent, what is the total heat loss? Starting from the point 0.6 on the scale at the bottom of figure 42, follow the vertical line to its intersection with the lowest curve; then from this point follow the horizontal line to the scale at the left, which gives the heat loss as 3 per cent of the total heat in the coal.

If the gases contain 10.3 per cent of CO , and 0.8 per cent of CO , then the vertical line representing 0.8 per cent CO is followed to the highest curve, and from this intersection point a horizontal line is followed to the scale at the left, which indicates the heat loss to be 5.1 per cent of the total heat in the coal. Similarly, if the content of CO_2 in the furnace gases is 12.2 per cent the middle curve is used for the determination of the heat loss.

TEMPERATURE DROP ALONG THE PATH OF GASES IN THE LONG COMBUSTION CHAMBER.

Column 17 of Table 3 shows that there is a steady temperature drop along the path of gases from section A to section G. This temperature drop is mainly due to the following causes:

- (a) Some heat is absorbed from the products of combustion by the water in the gas-sampling tubes.
- (b) Some heat is radiated from the walls and roof of the furnace.
- (c) Some heat is radiated to the opening under the boiler.
- (d) The temperature measured at section A is too high because the pyrometer was sighted on the flames, which were at a higher temperature than the gases.

HEAT ABSORBED BY WATER IN GAS SAMPLERS.

The heat absorbed by the cooling water in the gas samplers is by far the greatest cause of the temperature drop from A to G as shown by actual determination in test 179. The weight of water circulated through the samplers per hour as measured by a meter was 34,760 pounds. The rise in temperature of the water due to the absorption of heat from the furnace was 44 F. The total heat absorbed and carried away by the water was $34,760 \times 44 = 1,529,440$ B. t. u. The weight of gases passing through the furnace per hour, figured from the gas analysis and the rate of combustion, was 28,320 pounds. If the average specific heat of gases at the furnace temperature be taken as 0.2774, the heat given up by the gases for each degree of temperature drop is $28,320 \times 0.2774 = 7,856$ B. t. u. Therefore the temperature drop caused by the loss of 1,529,440 B. t. u. to the gas samplers is $1,529,440 \div 7,856 = 195^\circ \text{ F.}$ or 109° C.

The high rate at which these samplers absorbed heat is worthy of note. The quantity of heat absorbed by the samplers per hour is equivalent to a rate of heat absorption of $1,529,440 \div 33,300 = 46$ boiler horsepower.

There were 16 samplers, each $1\frac{1}{2}$ inches in outside diameter and 23 inches in length, exposed to the furnace gases. This gives a total heating surface of a little under 10 square feet. Each square foot of the heating surface transmitted an amount of heat equivalent to 4.6 boiler horsepower, or one horsepower was generated on less than 0.22 square foot of heating surface.

These figures compare favorably with the figure 0.357 square foot of heating surface per boiler horsepower, which is the rate of heat transmission of the lowest row of boiler tubes at the point where the hot products of combustion first come in contact with the boiler, as determined in tests described in Technical Paper 114.^a This rate was computed from the measured temperature drop through the metal of the tube. The difference between the two figures is in the right direction. It stands to reason that the samplers absorb more heat than boiler tubes because the temperature in the furnace is higher and more heat is imparted to them by radiation than to the boiler tube.

RADIATION LOSSES FROM WALLS AND ROOF OF FURNACE.

In furnace tests described in Bulletin 8,^b it is shown that about 465,600 B. t. u. is radiated from the walls and the roof of the furnace per hour. This radiation loss reduces the temperature of the gases by $465,600 \div 7,856 = 59^\circ \text{ F. or } 33^\circ \text{ C.}$

The heat absorbed by the cooling water in the gas samplers and the heat radiated account for 141° C. of the temperature drop of the gases at the end of the combustion chamber. There is still a drop of about 59° C. to be accounted for.

HEAT RADIATED TO OPENING UNDER BOILER.

Part of this unaccounted for temperature drop of 59° C. is due to heat radiation from the fuel bed and flames to the opening under the Heine boiler. The surfaces of the space under the boiler are at a considerably lower temperature than the flames at section A of the furnace. On account of this temperature difference the flames impart heat by radiation to these surfaces. The radiation is both direct, as shown by the straight line in figure 43, and reflected, as shown by the zigzag lines. In the latter instance the heat is radiated

^a Kreislinger, Henry, and Barkley, J. F., Heat transmission through boiler tubes: Tech. Paper 114, Bureau of Mines, 1915, p. 25.

^b Ray, W. T., and Kreislinger, Henry, The flow of heat through furnace walls: Bull. 8, Bureau of Mines, 1911, 32 pp.

from wall to wall owing to the difference of temperature between successive portions of the inner surfaces. In figure 43 these descending temperature steps are indicated by T_1 , T_2 , T_3 , and T_4 . Of course, in reality there are no such sudden steps; the high temperature gradually shades down through an infinite number of intermediate temperatures. The heat also travels in a similar zigzag manner by actual reflection from the surfaces as light does from smooth objects. It is difficult to estimate the quantity of heat transmitted from the flames to the surfaces under the boiler, and the corresponding temperature drop due to it; probably it does not exceed 25°C .

TEMPERATURE OF FLAMES AT SECTION A
HIGHER THAN AVERAGE TEMPERATURE
OF GASES AT THE SAME SECTION.

As the particles of combustible burn, the heat generated by their combustion raises the temperature of the particles immediately surrounding those in the process of combustion above those farther away, and time is required before the temperature is equalized. Thus, the flames contain centers of combustion with temperatures higher than the average temperature of the gases at the same cross section. When an optical pyrometer is used for measuring the temperature, the instrument unavoidably covers with its field some of these centers of intense combustion and gives a reading somewhat higher than the average temperature. It is difficult to estimate the error due to this cause, but in most of the measurements it probably is not over 50°F .

The temperature drop due to the first three causes mentioned is less as the rate of combustion increases. Thus comparison of figures 11 and 12 show that in test 179 having a rate of combustion of 59 pounds the drop is much less than in test 173 having a rate of combustion of 39 pounds.

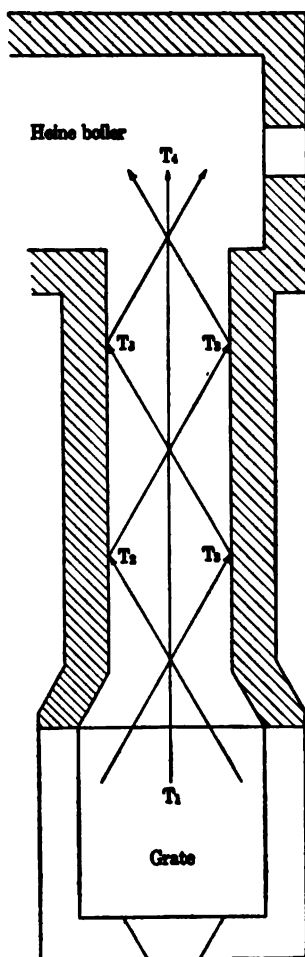


FIGURE 43.—Diagram showing temperature drop from radiation.

DISCUSSION OF THE PROCESS OF COMBUSTION ABOVE THE FUEL BED.

The weights of the combustible gases, tar, and soot in the furnace gases for a number of tests with Pittsburgh screenings are shown in Table 11 following:

TABLE 11.—Results showing weights of combustible gases, tar, and soot rising from fuel bed, in grams per cubic foot of gas at 60° F. and 30 inches pressure.

[All tests made with Pittsburgh screenings.]

Test No.	Rate of combustion, pounds.	Pounds of air per pound of combustible fired.	Distance of sample from fuel bed, ^a	Combustible gases, grams per cubic foot.					Tar, grams per cubic foot.	Soot, grams per cubic foot.	Total soot and tar, grams per cubic foot.	Total combustible, grams per cubic foot.	Soot and tar, per-centage of total combustible.
				C in CO.	CH ₄ .	H ₂ .	Unsaturated hydrocarbons.	Total gaseous combustible.					
1	2	3	4	5	6	7	8	9	10	11	12	13	14
195	24.0	11.8	0	2.211	0.105	0.140	0.035	2.491	0.054	0.180	0.234	2.725	8.6
			11	1.250	.071	.071		1.392	.010	.169	.179	1.571	11.4
			A	.668	.036	.036		.740	.004	.128	.132	.872	15.1
207	21.4	17.8	0	1.197	.071	.071	.142	1.481	.723	.301	1.024	2.505	40.9
			11	.489	.036	.018		.543	.003	.089	.092	.635	14.5
			A	.340	.018	.014		.372	.001	.054	.055	.427	12.9
206	21.4	21.2	0	1.157	.071	.036	.035	1.299	.020	.064	.084	1.383	6.1
			11	.605	.018	.018		.641	.005	.075	.080	.721	11.1
			A	.402	.018	.014		.434	.003	.059	.062	.496	12.5
185	22.1	22.8	0	.383		.013		.396			.063	.459	13.7
			11	.152		.004		.156			.027	.183	14.8
			A	.030				.030					
196	32.0	11.2	0	3.417	.068	.102	.034	3.621	.019	.138	.157	3.778	4.2
			11	1.063	.036	.072	.036	1.207	.019	.220	.239	1.446	16.5
			A	1.436	.036	.036		1.508	.008	.121	.129	1.637	7.9
190	31.3	14.8	0	1.466	.314	.140	.140	2.060	.342	.198	.540	2.600	20.8
			11	.595	.018	.036		.649	.008	.190	.198	.847	23.4
			A	.095		.004		.099	.004	.156	.160	.259	61.8
186	27.2	15.7	0	.746	.071	.036	.036	.889			.115	1.004	11.5
			11	.526	.018	.014		.558			.070	.628	11.1
			A	.203		.018		.221					
191	29.5	10.6	0	1.708	.018	.035		1.761	.020	.072	.092	1.853	5.0
			11	.886	.018	.025		.929	.011	.174	.185	1.114	16.6
			A	.047		.011		.058	.006	.060	.066	.124	53.2
203	33.1	22.4	0	1.774	.106	.106	.071	2.057	.042	.177	.219	2.276	9.6
			11	.485	.018	.014		.517	.003	.058	.061	.578	10.6
			A	.978	.018	.057		1.053	.003	.093	.096	1.149	8.4
198	40.5	11.4	0	1.363	.035	.069	.069	1.536	.026	.091	.117	1.653	7.1
			11	1.437	.036	.106		1.578	.007	.227	.234	1.812	12.9
			A	1.235		.064		1.289	.003	.085	.088	1.377	6.4
197	39.2	12.6	0	1.480	.140	.105	.140	1.965	.409	.263	.672	2.537	26.5
			11	.871	.018	.036		.925	.006	.144	.150	1.075	14.0
			A	.847	.036	.054		.937	.004	.162	.166	1.108	15.0
188	38.0	13.6	0	1.743	.070	.070	.035	1.918	.107	.080	.187	2.105	8.9
			11	1.182	.071	.071		1.324	.013	.284	.297	1.621	18.3
			A	.464		.037		.601					
189	39.2	14.1	0	1.235	.071	.071		1.377	.024	.113	.137	1.514	9.0
			11	.595	.036	.036		.667	.014	.160	.174	.841	20.7
			A	.403		.036		.439					
187	35.6	14.5	0	2.505	.173	.208	.311	3.197	2.232	.272	2.504	5.701	43.9
			11	.955	.072	.072		1.099	.006	.069	.075	1.174	6.4
			A	.451		.036		.487					

^a In the results 0 indicates samples collected at surface of fuel bed, 11 indicates samples collected 11 inches from fuel bed, and A indicates samples collected at section A of the combustion chamber.

TABLE 11.—*Results showing weights of combustible gases, tar, and soot rising from fuel bed, in grams per cubic foot of gas at 60° F. and 30 inches pressure—Continued.*

Test No.	Rate of combustion, pounds.	Pounds of air per pound of combustible fired.	Distance of sample from fuel bed.	Combustible gases, grams per cubic foot.					Tar, grams per cubic foot.	Soot, grams per cubic foot.	Total soot and tar, grams per cubic foot.	Total combustible, grams per cubic foot.	Soot and tar, percentage of total combustible.
				C in CO.	CH ₄ .	H ₂ .	Unsaturated hydrocarbons.	Total gaseous combustible.					
1	2	3	4	5	6	7	8	9	10	11	12	13	14
193	42.0	14.5	0	1.843	.027	.052		1.916	.025	.089	.114	2.080	5.6
			11	1.125	.018	.036		1.179	.006	.143	.149	1.328	11.2
			A	.374	.018	.029		.421	.002	.102	.104	.525	19.8
205	35.2	16.2	0	2.261	.039	.084	.105	2.479	.093	.171	.264	2.743	9.6
			11	1.473	.060	.078		1.611	.027	.272	.299	1.910	15.7
			A	1.083	.018	.025		1.126	.008	.050	.058	1.184	4.9
194	44.2	19.0	0	1.377	.021	.060		1.458	.013	.052	.065	1.523	4.3
			11	.337		.011		.348	.071	.073	.421	1.910	17.3
			A	.061				.061	.001	.016	.017	.078	21.8
204	37.9	20.6	0	1.341	.039	.039	.035	1.454	.024	.073	.097	1.551	6.3
			11	.367	.018	.025		.410	.015	.145	.160	.570	28.1
			A	.358		.014		.372	.003	.033	.036	.408	8.8
199	47.1	12.6	0	2.072	.206	.240	.034	2.562	.056	.173	.229	2.781	8.2
			11	.476	.018	.050		.544	.003	.114	.117	.661	17.7
			A	.958	.018	.040		1.016	.003	.061	.064	1.080	5.9
192	45.3	14.8	0	2.543	.038	.063		2.664	.031	.057	.088	2.752	3.2
			11	.703	.018	.039		.750	.006	.111	.116	.866	13.4
			A	.646	.018	.029		.693	.004	.091	.096	.788	12.1
200	50.5	18.3	0	.617	.253	.090	.108	1.068	.079	.052	.131	1.199	10.9
			11	.083		.014		.107	.003	.055	.058	.165	35.2
			A	.493	.018	.025		.536	.002	.041	.048	.579	7.4
201	60.0	15.1	0	1.303	.078	.078	.035	1.494	.032	.112	.144	1.638	8.8
			11	.448	.018	.029		.495	.003	.104	.107	.602	17.8
			A	.559	.018	.029		.606	.002	.064	.066	.662	8.5
202	57.3	16.0	0	1.334	.077	.077	.035	1.523	.008	.056	.064	1.587	4.0
			11	1.036	.039	.060		1.125	.003	.133	.136	1.261	10.8
			A	.310	.018	.018		.346	.002	.045	.047	.393	12.0

The last column of Table 11 shows that of the combustible matter rising from the fuel bed, roughly 12 per cent is in the form of tar and soot. Immediately at the surface of the fuel bed the quantity of tar is large, but decreases rapidly as the gases pass through the combustion space. On the other hand, the soot increases during the first foot of the gas travel. This fact is shown in figure 44, compiled from tests made with Pittsburgh screenings. In the left half of the figure the tests are grouped according to rate of combustion regardless of air supply. In the right half of the figure the tests are grouped according to excess air supply regardless of the rate of combustion. The object of grouping the tests together was to avoid the plotting of too large a number of curves, and also to bring out better the general relation between soot and tar at the different places of sampling. The results of individual tests vary so much among themselves that the general tendency is not easily apparent. Furthermore, it was thought desirable to bring out the effect of the rate of combustion

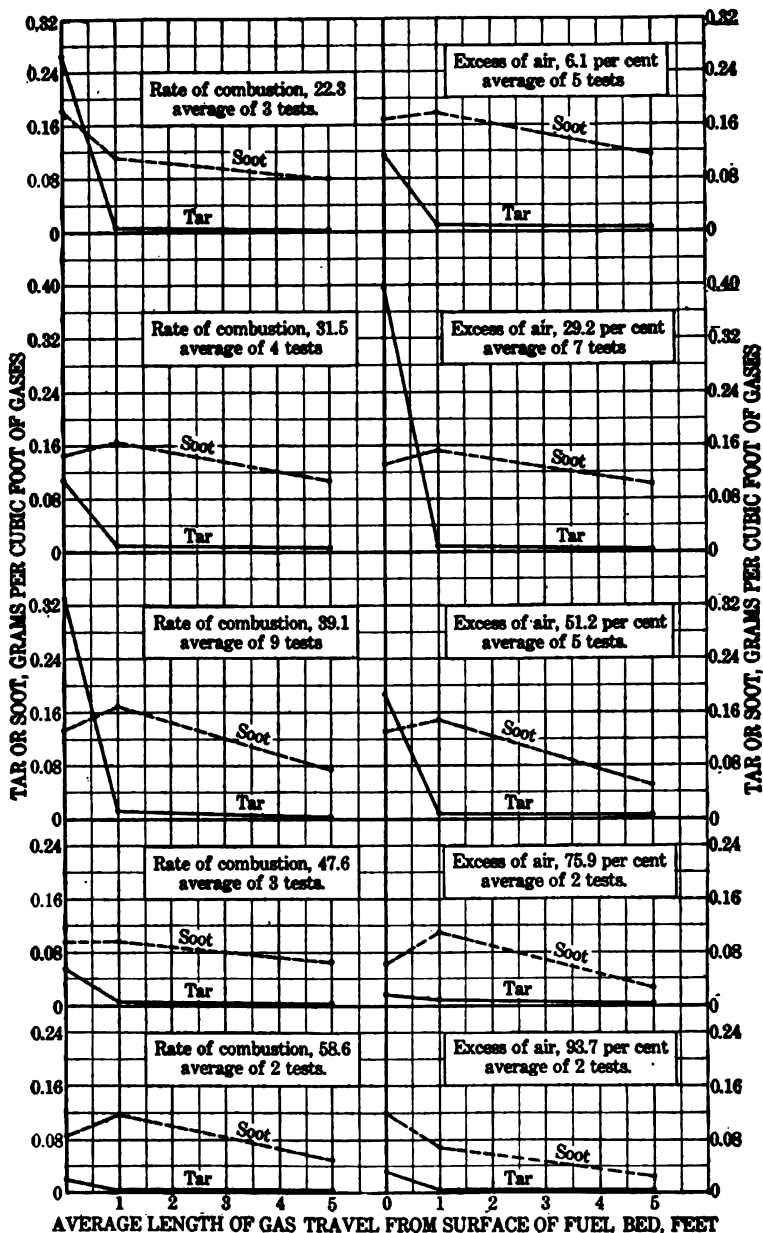


FIGURE 44.—Weight of soot and tar in furnace gases at the surface of, 1 foot from, and an average of 5 feet from the fuel bed. Tests made with Pittsburgh screenings. Tests at left are grouped according to rate of combustion regardless of air supply; those at right are grouped according to air supply regardless of rate of combustion.

and the excess of air on the quantity of tar and soot if any such effect existed. The abscissa in the figure is the length of gas travel from the grate, the ordinates are grams of soot or tar per cubic foot of gas. The weight of tar and soot was obtained from samples collected as described on page 20.

In general an increase in the rate of combustion and in the excess of air is accompanied by a decrease in the quantity of soot, and particularly in the quantity of tar. With all rates of combustion and all excess of air there is a large decrease in the quantity of tar and a moderate increase in the quantity of soot during the first foot of the length of gas travel. This last statement is true of four groups out of five. This decrease in the quantity of tars and increase in the quantity of soot seems to indicate that the volatile matter leaves the fuel bed as heavy hydrocarbons mostly in the form of tars. These tars are decomposed by the high furnace temperature and the absence of oxygen into soot and lighter, mostly gaseous hydrocarbons. The process of the decomposition of the hydrocarbons very likely consists of a number of consecutive reactions, each step of which is accompanied by the deposition of soot and formation of lighter hydrocarbons. This process of decomposition is complicated by the presence of CO_2 , which reacts with soot and combustible gases and is itself reduced to CO . Some of the possible reductions are indicated by the chemical equations on page 124. The decomposition and reduction proceed toward the simple gases CO and H_2 . This view is supported by the fact that samples collected at section A contain only a trace of tars and CH_4 , and a considerable percentage of CO and H_2 . Of course, the process of combustion takes place along with the process of decomposition as far as the supply of oxygen and the mixing makes combustion possible. However, it would be difficult to account for the absence of tarry and gaseous hydrocarbons and the high percentages of CO and H_2 at section A without decomposition and reduction; for, if combustion alone is considered, the simple gases CO and H_2 would burn first, leaving the heavier compounds to be consumed last.

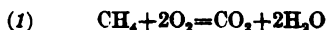
The length of time in which the tars are decomposed into soot and gases is very short. At the rate of combustion of 30 pounds per square foot of grate per hour the gases travel with a velocity of about 10 feet per second. As most of the tars disappear during the first foot of the gas travel from the fuel bed, the time taken for the decomposition of the tar is about one-tenth of a second. This high rate of decomposition is undoubtedly due to the high temperature near the fuel bed, which in the tests was probably not less than 1500°C .

SMOKE IS FORMED AT OR NEAR SURFACE OF THE FUEL BED.

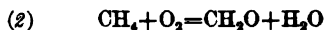
In the light of the preceding discussion it appears that the soot, which is the main constituent of visible smoke, is formed at, or very near, the surface of the fuel bed and not at the place where the furnace gases strike the heating surfaces of the boiler. The heating surfaces merely cool the gases surrounding the soot, thereby preventing its combustion. The formation of soot at the surface of the fuel bed is caused by the high furnace temperature and absence of oxygen. It is possible that if oxygen was present in sufficient quantity at the time of distillation of volatile matter, the heavy hydrocarbons would burn directly to products of complete combustion, CO_2 and H_2O , without first decomposing and depositing soot.

The different processes that may take place according to whether the volatile matter is distilled in the presence of different concentrations of oxygen or in its entire absence can be illustrated by the known behavior of methane at high temperature.

In the presence of a sufficient supply of oxygen methane burns completely, according to the equation:



With insufficient supply of oxygen partial combustion and decomposition of methane take place, which processes are represented by the equations:



In complete absence of oxygen only the process expressed by equation 3 takes place. The carbon precipitates as soot.

With the heavy hydrocarbons the effect of the concentration of oxygen is probably more pronounced because of the great multiplicity of simpler compounds into which they can be decomposed.

After the soot has once been formed it is difficult to burn it in the atmosphere of the furnace. This fact has been observed by many investigators, and some of the early writers on combustion even considered soot as noncombustible. At present no support can be found for this extreme view, as it can be easily shown that soot burns readily if oxygen is furnished in highly concentrated form. For example, if soot is mixed in the right proportion with potassium chlorate and the mixture thrown on a porcelain plate heated to about 500°C ., the soot burns almost instantaneously with a bright flash. Also, if soot is placed in a bomb calorimeter and surrounded with an atmosphere of compressed oxygen, and a source of ignition applied, it burns completely in a very short time.

CAUSES OF THE SLOW BURNING OF SOOT IN A FURNACE.

The soot burns very slowly in the furnace gases because the oxygen is greatly rarified, the gases containing only a few per cent of free oxygen; it burns rapidly in the bomb calorimeter or in mixture with potassium chlorate because it is surrounded with highly concentrated oxygen ready to combine with it.

As a matter of fact, all combustible substances burn slowly in an atmosphere of highly diluted oxygen, but in the case of soot this slowness is much more pronounced. The reason for the very slow combustion of soot in highly diluted oxygen probably lies in its complex molecular structure. It is commonly supposed that a molecule of soot consists of a considerable number of atoms, and that a similarly large number of molecules of oxygen is required to come in contact with the molecule of soot before the latter can combine with the oxygen. The chances of the molecule of soot finding this large number of molecules of oxygen in the furnace gases are small, and hence the slow combustion. Thus assuming the molecule of soot to consist of 12 atoms and represented by the symbol C_{12} , there will be required 12 molecules of oxygen to burn the one of soot. The reaction for the combustion of the soot may be expressed by the formula $C_{12} + 12 O_2 = 12CO_2$. For comparison the reaction for the combustion of CO is given in the following equation, $2 CO + O_2 = 2 CO_2$.

It is easy to understand that the two molecules of CO can find one molecule of oxygen in the furnace gases much more quickly than one molecule of soot can find 12 molecules of oxygen. Therefore simple gases like CO and H_2 burn quickly in the furnace, because they more readily find the small amount of oxygen needed for their combustion; whereas the more complex combustible like soot burns slowly because it can not readily find enough oxygen to effect its combustion.

In general, it may be said that the more complex a molecule of combustible is, the slower is its combustion in any given concentration of oxygen.

At present nothing is known of the molecular structure of soot; the number 12 has been taken only for illustration. There are indications that the actual index number is higher than 12 rather than lower. Experience with burning soft coal shows that if soot is once formed a large percentage remains floating in the gases after all other gaseous combustible has been almost completely burned. In a hand-fired furnace analysis of the flue gases will sometimes show no CO when at the same time black smoke is coming out of the stack. Even a very large combustion space is not an unfailing solution for burning soot, without a large excess of air. In the experimental furnace hav-

ing a long combustion chamber, when a shovelful of coal was spread over the fuel bed the smoke traveled the full length of the combustion chamber and came out of the stack in spite of the apparent whirling motion of the gases and the high temperature in the combustion space.

REASON FOR SUCCESS OF MECHANICAL STOKERS IN BURNING SOFT COAL WITHOUT SMOKE.

It seems that most mechanical stokers are smokeless not because they burn the smoke, but because they burn the coal in such a way that very little soot or smoke is produced. Hand-fired furnaces are smoky because soot is produced in or near the fuel bed, and can not be burned in the limited combustion space of the furnace.

Two of the most important conditions that make mechanical stokers successful in burning smoky coal without smoke probably are: (a) Slow and uniform heating of the coal with the result that a large part of the volatile matter is distilled at low temperature, and (b) distillation of the volatile matter occurs in presence of oxygen.

The great advantage of uniform feeding of coal and air with mechanical stokers is pointed out and discussed in Technical Paper 80^a and is generally known, so that no space need be devoted to it in this discussion.

DISTILLATION OF VOLATILE MATTER AT LOW TEMPERATURE.

Distillation at low temperatures favors the formation of light hydrocarbons of the paraffin series which contain more hydrogen and less carbon than the hydrocarbons of the aromatic group, which are distilled at high temperature. The hydrocarbons of the paraffin series are more stable at high temperatures and on account of their higher hydrogen content are more apt to burn completely without depositing soot than the compounds containing more carbon. It requires one molecule of oxygen to burn completely one atom of carbon, whereas one molecule of oxygen burns completely four atoms of hydrogen. Thus of two compounds having the same number of atoms in a molecule, the one having more hydrogen requires less oxygen for its combustion, and therefore, in the same concentration of oxygen will burn more readily. Instead of the volatile matter being distilled as a heavy hydrocarbon of the composition, say, $C_{15}H_{12}$, and then reduced to $3 CH_4$ by dropping C_{12} , the volatile matter should be distilled as light hydrocarbons, such as CH_4 , and the carbon left in the fuel bed as fixed carbon, in which form it can be burned without any trouble to CO_2 or CO . In the atmosphere above the fuel bed it is

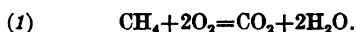
^a Kreislinger, Henry, Hand-firing soft coal under power-plant boilers: Tech. Paper 80, Bureau of Mines, 1915, p. 18.

much easier for a molecule of CH_4 to find two molecules of oxygen for its combustion than it is for a molecule of $\text{C}_{14}\text{H}_{10}$ to find its requirement of 18 molecules. The general formulas of the more common hydrocarbon compounds are given in the table following:

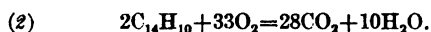
Some of the more common hydrocarbons arranged in order of decreasing contents of hydrogen.

	Compound.	Formula.
Unsaturated hydrocarbons.....	Paraffins.....	$\text{C}_n\text{H}_{2n+2}$
	Olefins (ethylene).....	C_nH_{2n}
	Acetylene.....	$\text{C}_n\text{H}_{2n-2}$
Aromatic hydrocarbons.....	Benzene.....	$\text{C}_n\text{H}_{2n-6}$
	Naphthaline.....	$\text{C}_n\text{H}_{2n-12}$
	Anthracine.....	$\text{C}_n\text{H}_{2n-18}$

In the above table attention is called to the decrease of the hydrogen contents as we pass from paraffins through the unsaturated and aromatic hydrocarbons to the anthracine. If these formulas be considered as purely arithmetical formulas it is plain that in the anthracines the n can not be less than 10, whereas in the case of paraffins n is as low as 1, which gives the formula of methane, CH_4 , which is the lightest hydrocarbon and burns more readily than any other carbon-hydrogen compound. A molecule of methane needs for its complete combustion two molecules of oxygen, as shown by equation 1 following:



On the other hand, the lightest anthracine imaginable would be C_{10}H_8 , which would take not less than 11 oxygen molecules. The lightest anthracine known is $\text{C}_{14}\text{H}_{10}$, which burns according to equation 2.



That is, to burn two molecules of anthracine would require 33 molecules of oxygen.

The preceding discussion shows the advantage of distilling volatile matter at low temperatures producing mostly paraffins or other hydrocarbons having light molecules. The furnace should be so designed that the distillation takes place at low temperature. After the volatile matter is distilled air should be added and the mixture then passed through a hot chamber.

DISTILLATION OF THE VOLATILE MATTER IN PRESENCE OF OXYGEN.

The principal methods of feeding coal and air in commercial furnaces are indicated in figure 45. In the hand-fired furnace (A) the air first comes in contact with the partly burned coal at the bottom of the fuel bed containing no volatile matter. Before the air reaches the distillation zone at the top of the fuel bed all the oxygen is practically used, so that the distillation occurs in absence of oxygen.

The heat is imparted to the freshly fired coal by conduction and by radiation from the hot fuel bed below, by convection from the hot gases passing through it, and by radiation from the hot furnace walls and fire-brick arches, if there happen to be any. Therefore the heating of the fresh coal is very rapid. In the side-feed types of stoker

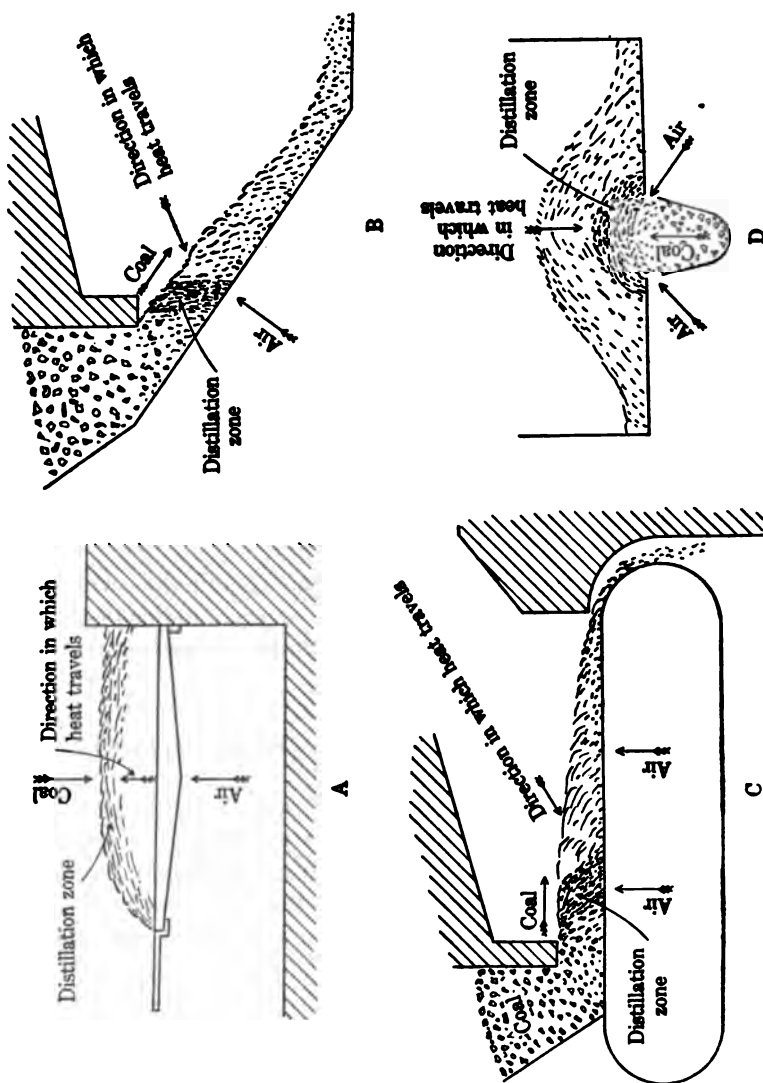


FIGURE 45.—Principal methods of feeding coal and air in commercial furnaces. *A*, hand feed; *B*, side feed by gravity and agitation of grate bars; *C*, side feed with chain grates; *D*, underfeed. The arrows indicate direction of feeding coal and air, and also the direction in which heat travels and is imparted to the fresh coal. The distillation zone is indicated by dark shading.

(B and C) the distillation zone extends the full thickness of the fuel bed, and the air comes in contact with the fresh coal before the oxygen is all used, so that the distillation occurs in an oxidizing atmosphere. The heat is imparted to the fresh coal by conduction and radiation from the hot coal in front, and by radiation from the hot

furnace walls and arches. No heat is imparted by convection from the air passing through. Therefore the heating is slower than in a hand-fired furnace. In the underfeed type of stoker (D) the air and coal is fed into the furnace in the same direction, so that the distillation occurs in air with its content of oxygen previously undiminished by combustion. The same is true in the burning of pulverized coal. In the underfeed stoker the heating of the coal takes place only by conduction and radiation from the upper layers of the fuel bed, against the convection from the stream of cold air. The heating is therefore slow.

A study of figure 45 will show that in most of the common types of mechanical stokers the distillation of volatile matter occurs in the presence of oxygen, whereas in hand-fired furnaces the distillation takes place in almost entire absence of oxygen, all of the latter being consumed as it passes through the fuel bed.^a Even if there should become tendency to decomposition with the mechanical stokers, the presence of large percentages of oxygen at the point of distillation makes it possible for the hydrocarbons to react with oxygen before the deposition of carbon can really take place.

High temperatures such as exist in boiler furnaces and absence of oxygen promote the decomposition of all hydrocarbons, including methane, the lightest of the paraffin series, one of the products of decomposition being soot.

When methane is burned with insufficient air supply, it burns with a yellow flame and deposits soot. Therefore, natural gas, which consists mostly of methane, should be burned with some excess of air and with provision for obtaining a good burning mixture; otherwise soot will be deposited and gas will be wasted, owing to incomplete combustion. In this respect natural gas differs greatly from producer gas, the latter consisting mostly of carbon monoxide, which does not readily decompose. An instance showing the difference in the behavior of the two gases came recently to the writer's attention. A porcelain-ware manufacturer used producer gas for burning his product. The kiln in which the burning was done had one chamber for the ware and another surrounding it for the burning of the gas, the two chambers being separated by a partition of a highly refractory material. The products of combustion did not come in direct contact with the ware, so that the heat had to be transmitted through the partition. The gas to be burned was introduced into the combustion chamber all at one point, but to have the temperature more uniform throughout the chamber, air was admitted at several points along the path of the combustion gases. Thus the process of combustion was distributed over a large space and nearly uniform temperature was obtained.

^a Kneisinger, Henry, Ovits, F. K., and Augustine, C. E., Combustion in the fuel bed of hand-fired furnaces, Tech. Paper 137, Bureau of Mines, 1916, 76 pp.

This method of obtaining uniform temperature worked splendidly with producer gas. The owner was persuaded to change to natural gas, and when he tried the same method of burning it as he used with the producer gas, he found that a layer of soot was deposited on the partition, preventing to a large degree the transfer of heat to the ware. At the same time a large part of the heat value of the carbon in the methane was never developed, owing to deposition as soot. He used entirely too much gas and obtained unsatisfactory results.

The amount of air admitted at the point where the gas was introduced, which was by design far from enough for complete combustion, was sufficient to heat the methane to a temperature at which it decomposed into carbon, hydrogen, and some carbon monoxide. A large part of the carbon settled on the surface as soot while the hydrogen and carbon monoxide were burned with the air added along the path of the gases.

When changing to natural gas the method of burning it should also have been changed. The gas should have been introduced into the kiln in a number of small jets evenly distributed over the area where uniform temperature was desired. With each gas jet a sufficient amount of air should have been supplied to insure complete combustion and thus prevent the deposition of soot.

This affords a very instructive example of what happens when the hydrocarbons are burned with insufficient air supply. By air supply is meant the air available for combustion at the point where the hydrocarbons are heated or generated in the furnace; any air that is added later along their path, is added too late, because the hydrocarbons are already decomposed by heat into soot and more stable gases. To insure rapidity and completeness of combustion in burning soft coal the distillation of volatile matter, which consists mainly of hydrocarbons, should take place in the presence of oxygen in order to prevent the formation of soot. In most mechanical stokers this condition is more or less closely approached; in hand-fired furnaces it is not. As shown in Technical Paper 137* no oxygen penetrates to the distillation zone in hand-fired furnaces, hence the decomposition of the hydrocarbons in presence of CO, into soot, CO and H_2 .

The two conditions for smokeless combustion with mechanical stokers could be perhaps applied to gas producers to eliminate soot and tar from the producer gas; that is, to distill the volatile matter at low temperatures and in strongly oxidizing atmospheres so that the hydrocarbons burn to the products of complete combustion, CO, and H_2O ; then by passing these products through a bed of hot coke to decompose them into CO and H_2 . This process of gasification is similar to that which takes place in the underfeed type of mechanical stoker.

* Kreisinger, Henry, Ovitz, F. K., and Augustine, C. E., combustion in the fuel bed of hand-fired furnaces: Tech. Paper 137, Bureau of Mines, 1915, 76 pp.

STABILITY OF HYDROCARBONS.

Hydrocarbons are decomposed or cracked by heat. Carbon and hydrogen are the ultimate products when cracking is carried to the end. The decomposition is a complicated process, the exact mechanism of which is not known, but it is governed by the laws of physical chemistry. Equilibrium conditions determine the extent to which cracking occurs, and the values of the equilibrium constant depend, among other factors, upon the temperature and pressure. High temperature favors cracking because most of the reactions take place with absorption of heat. The influence of pressure need not be considered here as the pressure in boiler furnaces is always very nearly atmospheric.

Mayer and Altmayer investigated the stability of methane and found the following percentages were stable in the presence of hydrogen.

*Percentage of methane in equilibrium with hydrogen and carbon at various temperatures.**

Temperature, C.....	250°	450°	550°	750°	850°
CH ₄ , per cent.....	95.79	76.80	45.60	6.02	1.59

The reaction is represented by the expression $\text{CH}_4 = \text{C} + 2 \text{H}_2$. At 850° C. and atmospheric pressure 1.59 per cent of methane is in equilibrium with hydrogen whose partial pressure is then 0.9841 atmospheres. As the partial pressure of the hydrogen decreases the equilibrium pressure of methane decreases also. The partial pressure of hydrogen in the furnace is small and the temperature much higher than 850° C., therefore it is evident that only a very small percentage of methane can exist in equilibrium in a boiler furnace. This deduction is confirmed by the fact that only small traces of CH₄ were found in the long combustion chamber, except immediately at the surface of the fuel bed before it had had sufficient time to decompose.

Such equilibrium conditions as given for methane are not available for other hydrocarbons present in furnace gases, but experimental data show that methane is much more stable than ethylene or ethane. In general, the simple hydrocarbons are more stable than those of higher molecular weight.

Equilibrium conditions, although they tell the extent to which cracking will proceed if allowed sufficient time, do not give any information as to the velocity of decomposition, which is important in furnace and gas-producer practice.

The process of decomposition of hydrocarbons is probably a gradual dehydrogenation. Compounds of higher molecular weight are decomposed into compounds of lower molecular weight with liberation of hydrogen or a simple hydrocarbon. These intermediate

* Haber, F., *Thermodynamics of technical gas reactions*. Translation by Lamb, A. B., 1908, p. 245.

compounds are in turn decomposed. Some of the products, particularly the carbon, may polymerize to form compounds of higher molecular weight. The decomposition does not follow any single path, but is influenced by the temperature and pressure. The ultimate products are carbon and hydrogen.

Black smoke is the result of this decomposition of hydrocarbons driven from coal as volatile matter. The black part of smoke consists almost entirely of finely divided carbon. Analyses of the solid portion, or soot, show it to be nearly pure carbon, with a little tar and ash.

The temperatures in a boiler furnace are much higher than those given in the preceding table and therefore more rapid decomposition would be expected. Gases collected above the fuel bed of the furnace described in this bulletin contained only a small percentage of methane. At an average distance of 5 feet from the fuel bed one or two tenths of a per cent of methane was frequently found, which is about the limit of accuracy of the methods of gas analysis used. Unsaturated hydrocarbons which were found in the gas samples collected at the surface of the fuel bed had entirely disappeared at an average distance of 5 feet, and in most cases none were found 11 inches above the fuel bed.

Although the direct decomposition of hydrocarbons by heat may be a slow process, methane especially being decomposed slowly, their disappearance in the furnace may be quickened by their reaction with oxygen or some of the products of combustion such as CO , and H_2O . The following equations indicate some of the reactions that may occur:



Only when combustion is very poor, owing to inadequate space above the fuel bed and rapid cooling of the gases, are hydrocarbons likely to be found in flue gas.

What has been said regarding the decomposition of hydrocarbon gases applies also to the cracking of tar. Tar is a complex substance containing hydrocarbons of high molecular weight, which are liquid or solid at ordinary temperature.

Lunge^b gives a list of over 200 compounds which have been found or may reasonably be presumed to exist in coal tar. In contrast with gaseous hydrocarbons those which are liquid and solid contain a much larger number of carbon atoms in the molecule and consequently deposit more carbon when they are cracked.

^b Lunge, G., *Coal tar and ammonia*, 4th ed., 1909, p. 160.

Tar exists in the furnace in the form of vapor, an ideal condition for cracking. The small globules present a large surface for absorption of heat from the gases and hot furnace walls, and are quickly heated to a high temperature, which favors the formation of carbon.

On account of the complex nature of tar a great many reactions are involved in its decomposition. In general, the cracking is similar to that of hydrocarbon gases but many more compounds are involved and the result is a very complicated equilibrium among a large number of hydrocarbons. Very little experimental data are available on equilibrium and the velocity of these reactions; however, the high temperature in the furnace and the fact that the tar is in a state of fine subdivision favor rapid cracking and the formation of large amounts of carbon. This view is supported by the results shown in figure 44.

Table 11 also shows that the greatest amount of tar is found with a larger proportion of soot. The content of tar in the gases decreases rapidly as the distance from the fuel bed increases; at an average distance of 5 feet the tar has very nearly disappeared.

The velocity of combustion of hydrocarbons is faster than velocity of decomposition; therefore, combustion will take precedence over decomposition. For this reason air supplied over the fuel bed should be admitted as near to the surface of the bed as possible and mixed with the hydrocarbons, so that they will be burned before they are decomposed by heat and form smoke, which is difficult to burn in the rarified oxygen of the furnace.

PHYSICAL CHEMISTRY OF COMBUSTION OVER THE FUEL BED.

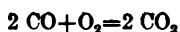
Combustion over the fuel bed is a chemical reaction between gaseous oxygen on the one hand and gaseous, liquid, and solid combustible, which rises from the fuel bed, on the other hand. The gaseous and liquid combustible results from gasification of fixed carbon on the grate and distillation of volatile matter. The solid combustible consists chiefly of finely divided carbon, known as soot, which is produced by decomposition of volatile matter. Also particles of coal are carried along with the current of gases. To burn these gaseous, liquid, and solid combustible materials oxygen must be supplied by admitting air over the fuel bed. The velocity and completeness of combustion are influenced by concentration, mixing, temperature, and time of contact of oxygen and combustible, and therefore a knowledge of their effects is important.

CONCENTRATION OR MASS ACTION.

According to the kinetic theory, gases consist of freely moving molecules. The contacts or collisions among molecules of different nature produce chemical action. Therefore for complete combustion

to take place in the mixture of gases, each molecule of the combustible must come in contact with the number of oxygen molecules required to burn it. The concentration or number of molecules of combustible gas and oxygen in a unit volume will influence the number of collisions taking place. According to the law of mass action, the velocity of a chemical reaction in the combustion process is proportional to the products of the concentrations of the oxygen and the combustible substances; that is, if one molecule of substance A reacts with one molecule of substance B the velocity of the reaction at any moment is proportional to the product of the number of molecules of A and B present in unit volume. If several molecules of one substance are taking part in the reaction, the concentration of that substance must be raised to the corresponding power. Thus if one molecule of substance A reacts with two molecules of substance C the velocity of the reaction will be proportional to the number of molecules of substance A in unit volume multiplied by the square of the number of molecules of substance C in unit volume. Thus it can be said that the velocity of a chemical reaction increases when the concentration of the reacting substances increases.

In the combustion of coal the law of mass action manifests itself by the fact that in boiler furnaces a considerable excess of air above the theoretical requirement is necessary to insure complete combustion within the furnace. If the air supply is reduced to nearly the theoretical amount the combustion is so slow that a large percentage of the gases may leave the furnace before they are burned. With low excess of air the time required to complete the combustion is long and the combustion space must be large. This fact is shown by the positions of the curves representing the different excesses of air in figures 29 and 30. In combining with the oxygen, carbon monoxide forms carbon dioxide according to the following reaction:



It can be easily seen that if there are two molecules of carbon monoxide and 10 molecules of oxygen in a unit volume, the chances of one molecule of oxygen meeting two molecules of carbon monoxide are greater than if there were only two molecules of oxygen present. When oxygen in a mixture of air and combustible in the furnace unites with carbon monoxide, this oxygen is no longer available for oxidation of the remaining combustible gas. Furthermore, the molecules of oxygen and combustible gas removed are replaced by the products of combustion which dilute the mixture, and the remaining molecules of oxygen and combustible gas come in contact less frequently. As combustion proceeds the concentration of the oxygen and combustible gas becomes less and less and the velocity of combination becomes slower and slower.

When dealing with the law of mass action the concentration of the chemical substances is expressed in gram molecular units. The volumetric weights of gases are directly proportional to their molecular weights, because equal volumes of all gases at the same temperature and pressure contain the same number of molecules. Therefore, with gases analyzed volumetrically all that is necessary is to substitute the volumetric percentages in the equation corresponding to the particular reaction and obtain the relative rate of combustion, assuming the same temperature and pressure in each case. For example, two samples of gas collected at section C of the combustion chamber showed the following percentages of O_2 and CO, the temperature in each case being $2400^\circ F.$:

- (1) $O_2=4.0$ per cent, $CO=0.5$ per cent.
 (2) $O_2=7.0$ per cent, $CO=0.4$ per cent.

The rest of each sample was nitrogen, carbon dioxide, and water vapor. What was the relative velocity of combustion of CO at each of the two sections? The equation representing the reaction between oxygen and carbon monoxide is $2 CO + O_2 = 2 CO_2$. At a temperature of $2400^\circ F.$ the dissociation of CO_2 is only about 0.2 per cent.

Therefore one may say with close accuracy that the reaction at this temperature is not reversible but proceeds to a finish from left to right. The relative rates of combustion in the two cases are:

- (1) $4.0 \times 0.5^2 = 1.0$.
 (2) $7.0 \times 0.4^2 = 1.12$.

The velocity of combustion in the second case is greater on account of higher concentration of CO.

Another example is given in the following: Assume that the gas mixture in the long combustion chamber contains at section A 10 per cent CO and 9 per cent O_2 , and at section C 0.6 per cent CO and 4.0 per cent O_2 , what are the relative rates of combustion of CO in each case? The rate of combustion of CO at section A is $9 \times 10^2 = 900$, whereas the rate of combustion of CO at section C is $4 \times 0.6^2 = 1.62$. These figures show that the rate of combustion of CO is over 500 times as fast at section A as it is at section C.

The following table gives the CO and O_2 content in the furnace gases sampled in test 187 at four sections of the path of gases, and the corresponding relative velocities of combustion calculated as in the two preceding examples. The four sections are: The surface of the fuel bed, 11 inches from the surface of fuel bed, section A, and section C. The calculated relative velocities are shown in figure 13 (p. 54) by the dotted curve which is nearly parallel to the curve of total combustible. With this dotted curve the scale at the right of the figure should be used.

* Nernst, W., and Wartenberg, H. von, Ueber die dissociation der Kohlensäuren: Ztschr. Phys. Chem., Bd. 58, 1906, p. 548.

Content of CO and O₂ in furnace gases of test 187 and the corresponding relative velocity of combustion of CO.

Point of sampling.	CO in gases.	O ₂ in gases.	Relative velocity of combustion of CO.
	<i>Per cent.</i>	<i>Per cent.</i>	
At surface of fuel bed.....	16.4	1.3	350
11 inches from fuel bed.....	6.4	2.9	119
Section A.....	3.1	3.2	31
Section C.....	0.4	4.9	0.78

When several combustible gases with more complex molecular structure are present, the calculation of velocities of reactions becomes complicated by reactions among the combustible gases themselves. The reactions are interrelated and can not be considered independently, but with simple gases such as in CO and H₂, less complication from side reactions is to be feared than with complex substances. At any rate the law of mass action is a valuable guide in such problems, and if the facts do not fit proved formulas well, usually too little theory has been applied, and some refining corollaries must be added.

With more than one combustible gas of simple molecular structure, the combined velocity of combustion can be obtained by adding together their individual velocities. However, care must be taken not to add these velocities without first multiplying each by the proper constant for that particular gas. At present not all of these constants are known, therefore it is not feasible to formulate an expression for the comparative total velocities of combustion of all the constituents taken together at several points along a flame.

Thus far the law of mass action as applied to the combustion of gases has been considered. This law is also a valuable guide in studying the combustion of the small particles of solid carbon, or soot, and the tar vapors, which are present as small globules. The mass, or concentration, of the solid carbon or the tar globule is very large compared with that of gaseous oxygen, so that the velocity of reaction depends on the rate with which the oxygen comes in contact with the surface of hot particles of carbon or tar globules. The faster oxygen is supplied to the surface, the higher is the velocity of the reaction. The rate at which oxygen is supplied to the surface of these carbon particles and tar globules is proportional to the percentage of the oxygen in the furnace gases.

The particles of carbon and the tar globules are so small that unless they are constantly agitated they tend to adopt the motion of the surrounding gas and consequently there is little or no relative velocity between them and the gas, to scrub off the layer of products of combustion and allow oxygen to act upon the surface of the

particles. Removal of the products of combustion from the surface and the supplying of free oxygen must be largely accomplished by natural diffusion, which is a slow process. This is one of the reasons why the combustion of soot and tar vapor is slow.

MIXING OF AIR AND GASES.

A chemical action can take place only when the reacting substances are brought into contact; this is true of gases as well as of liquids and solids. Thus, in the combustion of gases, which is a chemical process, there must be contact between the molecules of the combustible gas and of the oxygen before there can be combustion. This contact takes place most readily when the gases and the air containing the oxygen form a uniform mixture. The preceding calculations of the velocities of combustion according to the law of mass action are directly applicable only to uniform mixtures of the combustible gas and the air containing the oxygen. If they do not form a uniform mixture more time is required for the oxygen to come in contact with the combustible substances and the velocity of combustion will not be according to the law of mass action, but will be limited to the rate of mixing.

For example, if a cubic foot of CO and a cubic foot of O₂ were mixed thoroughly and the mixture placed in a hot furnace, combustion would proceed according to the law of mass action and would be rapid. If, on the other hand, the cubic foot of CO and the cubic foot of O₂ were placed in the hot furnace in such a way that the cubes would be in contact on one side only, combustion would take place only along the common surface of contact of the two gases and not within the cubes. The combustion would be slow, its velocity being limited to the rate at which molecules of CO and O₂ were coming to the common surface of contact. If this movement of molecules is left entirely to the natural diffusion of gases, the rate of the reaction would be slow—much slower than in a uniform mixture because natural diffusion of CO and O₂ is much slower than their velocity of combustion in uniform mixtures. If, however, the two cubic feet of CO and O₂ were violently stirred, the rate of CO coming in contact with O₂, or their mixing, would be rapid, and the velocity of combustion would approach that of uniform mixture, according to the law of mass action.

In commercial furnaces the combustible gases and the oxygen seldom form a perfectly uniform mixture, hence the rate of burning is not altogether controlled by the law of mass action but largely by the rate of mixing. This is especially true of the furnace gases near the surface of the fuel bed. The combustible gases leave the fuel bed in a solid stream containing little or no free oxygen. To this stream of combustible gas, air is added through the fire door or other special

openings. If this air is added in comparatively large streams that are not agitated, the streams of combustible gases and the air may flow through the furnace for some distance in more or less stratified streams, and the combustion take place only along the surfaces of contact and be slow. Such combustion makes itself evident as long flames. When the air is added in a large number of small streams, or when the large streams are agitated so that they break into a large number of small streams, the surface of contact is large and the combustion is rapid, approaching the velocity of combustion of a uniform mixture. Such combustion is manifested by short and hot flames.

Evidently, the length of the flame depends not only on the nature of the combustible, the excess of air, and the rate of firing, but also to a large degree on the rate of mixing of the combustible gases with the oxygen of the air. It has been shown in connection with Tables 8, 9, and 10 that the tendency of the gases is to flow in parallel streams even when the air was introduced into the furnace in many small streams through the tuyères of the furnace.

Mixing effects apply equally well to the combustion of soot and tar. The velocity at which these burn in the furnace depends upon the rate at which oxygen is supplied to their surfaces and the products of combustion are removed. Tar and soot particles are heavier than the gases surrounding them and change the direction of their motion less readily, therefore mixing creates a relative velocity between them and the gases. This relative velocity tends to scrub from their surfaces the products of combustion and to replace these with free oxygen.

Probably mixing has a further important effect on the combustion of the tar. A large part of the tar in the furnace is small globules, somewhat like visible steam rising from boiling water. When air is absent heating breaks up these globules of tar in various ways, the ultimate products being hydrogen and the finely divided solid carbon known as soot. In the presence of air it is quite likely that the reactions are different, oxygen taking part, and giving as final products CO , and H_2O with very little or no soot. This phase of combustion is discussed on pages 118 to 122.

A bunsen burner can be used to show how mixing air with a flame affects combustion. When the air supply is turned off the flame from a bunsen burner is luminous. Its luminosity is due to incandescent particles of solid carbon resulting from the decomposition of hydrocarbons in the gas. The interior of the flame contains no oxygen to burn the hydrocarbons, and they are decomposed by heat into hydrogen and carbon.

When air is mixed with the gas as it enters the burner, the flame is nonluminous and has a well-defined inner cone. In this inner cone the oxygen of the air combines with hydrocarbons of the gas, pro-

ducing H_2 and CO , which burn on the outer envelope of the flame. The reactions in the presence of oxygen produce gases that are easily burned, whereas the carbon produced in the absence of air is difficult to burn. Similarly, air mixed with volatile matter in the furnace before the tar is decomposed by heat may produce a larger proportion of combustible in the form of gas and less in the form of carbon.

Obviously, mixing is an important factor in combustion. Merely an excess of air in the furnace is not enough; the air must be supplied as near to the surface of the fuel bed as practicable, and be thoroughly mixed with the combustible matter in order that the combustion may be completed in a short time and a small combustion space made effective.

The importance of mixing was pointed out about 10 years ago by Breckenridge,* as follows:

On page 61 is given a calculation based on actual data obtained, and the conclusion is reached that the velocity of combustion decreases enormously from the surface of the fire to the rear of the combustion chamber, where it is relatively very small, the practical application is that little is to be gained by adding further length of smooth combustion chamber, which would be commercially as poor an investment of capital as to add to the length of a Corliss engine cylinder and stroke; we must resort to thorough mixing.

Breckenridge^b further states:

It will be probably found on attempting this reduction of dimensions and cost that the limit will be not in the boiler as such, but in the combustion chamber. Burning a large amount of coal in a small grate area is largely a question of draft and continual riddance of ash, but the rate of travel of gases through a combustion chamber is dependent, practically, only on the amount of carbon burned per unit of time, the rate being about the same no matter what the air supply per pound of carbon. As combustion chambers are now constructed for western coals they are too small, but "small" is a word not to be understood in this connection as referring to volume or length alone. By a "small combustion chamber" is meant a chamber in which either the time spent by the gas in traveling from the front to the rear of the boiler is short or the mixing devices are inefficient or absent. In the discussion of mass action it was stated that mere length of combustion chamber counts for little—that mixing is what counts—and thus there is a possibility of enormously increasing the efficiency of a combustion chamber as a burner of volatile matter. Efforts in completing a steam generating outfit of small dimensions must be largely concerned with the construction of a combustion chamber containing many gas-mixing appliances.

Mixing can be obtained by using special fire-brick structures in the combustion space, such as piers on the bridge wall or in the combustion space beyond the bridge wall, deflecting arches, and wing walls. Such structures change the direction of flow of the gases and create a relative velocity between tars and soot and the air, and between different streams of gases, thus tending to produce a uniform mixture. The chief objection to such structures seems to be

* Breckenridge, L. P., Study of four hundred steaming tests: Bufl. 325, U. S. Geol. Survey, 1907, p. 171.

^b Breckenridge, L. P., Work cited, p. 178.

their lack of durability. In these days of high furnace temperatures, the main condition of high efficiency, such structures are destroyed in a comparatively short time and their maintenance entails considerable expense.

Another method of mixing the gases and air is to introduce the air at high velocity either by means of steam jets or by forcing the air over the fuel bed with a blower. The jets of steam or of air cause a whirling motion over the fuel bed and tend to produce a uniform mixture. The best results are obtained when the air is admitted in a large number of small streams. When steam jets are used it should be remembered that it is not the quantity of steam but high-velocity streams of air that are wanted in the furnace. The steam jets are also inefficient as air movers.

TEMPERATURE IN COMBUSTION CHAMBER.

The velocity of chemical reaction increases with rise in temperature. At ordinary atmospheric temperatures the oxidation of carbon monoxide and hydrogen proceeds so slowly that the velocity of reaction can not be measured. At high temperatures and with thorough mixing, oxidation takes place rapidly; that is, the gases burn.

Experimental work has shown that at ordinary furnace temperatures the velocity is about doubled when the temperature is increased 10°C . At high temperatures the increase is less, but may be said to be of this order of magnitude. The velocity of reaction increases in geometric progression, whereas temperature increases in arithmetical progression. This means that if a reaction has a velocity of 1 at $1,000^{\circ}\text{C}$., assuming that the velocity is doubled for each increase of 10°C ., then at $1,100^{\circ}\text{C}$. the reaction has a velocity of 1,024, and at $1,200^{\circ}\text{C}$. the velocity is 1,048,576.

The temperature in a boiler furnace is about $1,400^{\circ}\text{C}$. At this temperature oxygen reacts with the combustible material to be burned very rapidly, but proper conditions for reaction must be provided.

When coal is burned carbon dioxide and water are the final products of complete combustion. The extent to which they are dissociated depends upon the temperature in the furnace and determines the limit of completeness of combustion.

Nernst and Wartenberg^a studied the reaction $2\text{CO}_2 \rightleftharpoons 2\text{CO} + \text{O}_2$, and found that the dissociation of CO_2 at one atmosphere at $1,323^{\circ}\text{C}$. was 0.104 per cent, at $1,423^{\circ}\text{C}$., 0.242 per cent, and at $1,523^{\circ}\text{C}$., 0.507 per cent.

Löwenstein^b studied the reaction for the formation of water, $2\text{H}_2\text{O} \rightleftharpoons 2\text{H}_2 + \text{O}_2$, and found that the dissociation at one atmos-

^a Haber, F., *Thermodynamics of technical gas reactions*, 1908, p. 323.

^b Haber, F., *Work cited*, p. 322.

phere at $1,432^{\circ}\text{C}$. was 0.102 per cent, at $1,510^{\circ}\text{C}$., 0.182 per cent, and at $1,590^{\circ}\text{C}$., 0.354 per cent.

As shown by the preceding data, at furnace temperatures carbon dioxide and water vapor are only slightly dissociated. Therefore it is evident that at equilibrium very little carbon monoxide and hydrogen can exist in the furnace. The combustion to carbon dioxide and water will be complete if given time enough.

TIME OF CONTACT.

If reactions are allowed sufficient time they will proceed to equilibrium. The length of time required depends upon the velocity of reaction; the slower the velocity the greater is the time necessary to reach equilibrium. As stated previously, CO_2 and H_2O are only slightly dissociated at furnace temperatures, and therefore when equilibrium is reached combustion is practically complete.

Equilibrium conditions determine what products are formed and the amounts. They do not give information on velocity of reaction. At present the velocity coefficients of the reactions that occur in the furnace are not all known definitely; therefore it is not possible to obtain an expression for their relative velocities. However, it is known that some of the combustible gases formed in the fuel bed will burn almost instantaneously at temperatures below those in the furnace when mixed with oxygen in proper proportions to make an explosion. An explosion is simply very rapid combustion, and therefore it can be said the velocity of combustion is sufficient to burn the gases in a very short time if they form a uniform mixture with a sufficient amount of air. In most furnaces that are in use the time required for complete combustion is not determined by the velocity of chemical reaction but by the rate of mixing.

One must remember that combustion of coal is not a simple oxidation of carbon and hydrogen to CO_2 and H_2O , but is a complex process. The volatile matter from heated coal contains tar, saturated and unsaturated hydrocarbons, carbon monoxide, and hydrogen, all of which must be burned in the space above the fuel bed. These materials all react among themselves and with the products of their combustion. Each reaction exerts its influence upon the others and can not be considered separately. The high temperature in the furnace favors the formation of simpler compounds so that the final products to be burned are largely soot, CO , and H_2 .

When other factors remain constant the velocity of combustion and the time of contact necessary for the completion of the reaction are closely related. The time of contact may be considered to be the length of time during which the combustible and the air supplied over fuel bed remain in the furnace. If the velocity is slow a long time of contact is required for complete combustion and a large

combustion space must be provided in order that the gases may be burned before they pass out of the furnace and are cooled. The rate of burning the coal and the size of combustion space determine the length of time the gases remain in the furnace. As the size of combustion space with a given furnace setting is constant the time the gases remain in the furnace is proportional to the rate of combustion. With high rates of combustion a larger volume of gases is produced and a larger volume of air must be supplied to burn it than with low rates of combustion. The larger volume of the mixture passes through the combustion space more rapidly; therefore the length of time the mixture stays in the furnace or the time of contact is shorter. The combustion space should be large enough so that the gases and the oxygen in the air supplied remain in contact long enough to burn the gases completely when the furnace is operated at its maximum capacity.

Concentration, mixing, temperature, and time of contact are interrelated and each of these factors is essential for good combustion. Consideration of these four factors indicate the conditions necessary to obtain good results. These conditions are: (1) Enough air; (2) air and the combustible gases thoroughly mixed; (3) a furnace temperature above the ignition point of the gases; (4) enough time for combustion to be complete.

In practice an excess of air is required, but the amount of excess air can be small if the mixing is thorough. Mixing is probably the most important single factor and the one most frequently responsible for poor combustion. A large excess of air and high temperature will do no good unless the air is mixed with the furnace gases and the oxygen and combustible gas brought into contact.

FUTURE METHODS OF USING BITUMINOUS COAL.

Difficulty in burning bituminous coal in industrial furnaces is due almost entirely to the volatile matter, because this leaves the fuel bed as gases and tars and must be burned in the combustion space of the furnace. Unless enough air is introduced immediately at the surface of the fuel bed and thoroughly mixed with the volatile combustible, the tars and the more complex combustible gases are quickly decomposed or "cracked" into soot and simple gases. The soot thus formed is difficult to burn in the rarified furnace atmosphere and is apt to pass out of the furnace as black smoke, particularly if the furnace is hand fired. To prevent smoke and its accompanying heat losses, a considerable excess of air must be used and the furnace must have a large combustion space containing gas-mixing devices. The fixed carbon is easy to burn because it stays on the grate. It burns partly to CO_2 , and partly to CO , which in turn can be burned to CO_2 with the additional air introduced above the

fuel bed. Practically complete combustion of CO is easily obtained because of its simple molecular structure.

During the past 15 or 20 years considerable attention has been given to designing of furnaces that would burn bituminous coals without producing smoke. Campaigns have been conducted to educate the fireman and induce him to use proper methods of firing the coal so as to produce the least amount of smoke. Large cities passed smoke ordinances aiming to reduce the smoke by encouraging or forcing the coal user to install apparatus that would burn coal smokelessly. These methods of attacking the problem have had much success but the production of unnecessary smoke continues. A large part of this smoke is made in a way that can not be easily controlled, even with good intentions on the part of the offenders.

In view of what is known of the chemistry of the various kinds of fuels and the possible advancement of such knowledge in the near future, it is questionable whether the method used in attacking the smoke problem was the best as regards fuel economy. The persistence of smokiness in burning bituminous coals shows that there is room for improvement in methods of burning, or that the best way of utilizing such coals has not been followed.

The volatile matter of bituminous coal would have greater economic value if converted into gas or liquid fuel than if burned under steam boilers. Under present market conditions heat in the form of coal brings 8 to 16 times the price of an equivalent amount of heat in the form of coal. Gas is an extremely convenient fuel and can be used to advantage for many purposes such as cooking, lighting and heating buildings, municipal lighting, and in some industrial plants for obtaining a uniformly high temperature and clean products of combustion. The residue from the coking coals should find a ready market for househeating and for steaming purposes. As both the gas and coke burn without smoke, their separate use would tend to make cities cleaner. The manufacture of gas from coal has been a commercial undertaking for many years and is still rapidly developing, the liberal margin between the price of heat in the form of gas and the price of heat in the form of soft coal making the conversion profitable.

The conversion of the volatile matter of bituminous coal into liquid fuel seems to be even more promising than its conversion into gas. By the application of proper processes it seems possible to reduce a large part of the volatile matter to liquid, of which an appreciable percentage can be obtained in the form of light oils suitable for motor fuel. Benzol has been obtained at by-product plants for many years without any special effort to produce it. There is no doubt that with well-developed methods the yield of benzol and similar oils could be greatly increased.

The demand for these oils is already great and will keep on increasing. The value of heat in the form of motor fuel is 20 to 30 times as great as that of heat in the form of coal. As the supply of bituminous coal is enormous, the uses of the oil are practically unlimited, and the margin of profit in the conversion is large, it would seem that the development of highly productive methods would be rapid. By itself the coke residue from such plants would have considerable commercial value, and if its price were made equivalent to that of coal it would doubtless find a wide market for househeating and steaming purposes.

Thus the use of coke and gas would gradually take the place of the use of bituminous coal as fuel and the smoke problem would largely disappear. By using coke under steam boilers they could be kept free from soot and the rate of heat transmission greatly increased. There are probably few instances where coke could not be substituted for coal.

The possibility of converting an appreciable part of the volatile matter of bituminous coal into motor fuel is indicated by Table 4 (p.78), which gives some of the chemical characteristics of the three coals that were tested in the experimental furnace. The part of coal that can be possibly converted into liquid fuel is the volatile carbon plus the available hydrogen. These two constituents form, on a moisture and ash free basis, about 13 per cent of the Pocahontas coal, 26 per cent of the Pittsburgh coal, and 34 per cent of the Illinois coal. The ratios of the volatile carbon to the available hydrogen for the three coals are 2.15, 4.36, and 6.6, respectively. The ratio of carbon to hydrogen in benzol (C_6H_6) is 12, the ratio of carbon to hydrogen in gasoline is about 5.3, computed from the average molecular formula of gasoline, C_8H_{18} . Evidently there is more than enough hydrogen for the volatile carbon to form benzol or gasoline. The necessary elements are there, the energy is there; the problem to be solved is to find a way to make the elements form the desired combination.

THE ADVANTAGES OF LIQUID FUEL.

The higher commercial value of liquid fuel is due to its being available for more purposes. Oil seems to have all the advantages of gas and solid fuel and none of their disadvantages. Gas, on account of its simple molecular structure, can be burned readily and without smoke in any commercial apparatus from a boiler furnace to a gas engine; also, the feed of gas can be made automatic for any apparatus and can be easily controlled. However, gas has the great disadvantage that it is not concentrated enough for convenient storage. On the other hand, coal contains large amounts of heat in highly concentrated form and can be easily stored. Its complex molecular structure and the many solid impurities tend to limit its use. In its

marketed form, coal lacks sufficient adaptability to delicate adjustment or automatic feed control.

Liquid fuel has a simple molecular structure, and the ease with which the light oils are changed to vapor make these oils readily adaptable to many purposes for which gas is used. Oil is a highly concentrated fuel and can be easily stored, transported, and handled.

CONCLUSION.

By such possible changes in the use of bituminous coal as are outlined in the preceding paragraphs, the problem of combustion will be much simplified.

Bituminous coal will probably continue to be used under steam boilers but to a lesser extent. In the future its principal use will be in the by-product plants because of the higher commercial value of the products that can be made from it. The higher the percentage of volatile combustible the higher will be the commercial value of the coal. The time may come when our views of the relative values of different coals will change and we shall consider anthracite as of minor importance as compared with the high-volatile bituminous coals.

Vague reports from Europe indicate that after the war the world will be informed of some extraordinary developments in the utilization of bituminous coal in certain countries, and that these developments will become of pressing importance to manufacturers in the United States.

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Bulletin 136

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FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

**DETERIORATION IN THE HEATING VALUE
OF COAL DURING STORAGE**

BY

HORACE C. PORTER AND F. K. OVITZ



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DETERIORATION IN THE HEATING VALUE OF COAL DURING STORAGE.

By HORACE C. PORTER and F. K. OVITZ.

INTRODUCTION.

Much has been written of the changes undergone by coal in storage and the deterioration of coal through exposure to the weather. In order to obtain definite information for the benefit of the Government departments and of all who store coal in large quantities, a series of tests was begun in the fall of 1909 under the supervision of J. A. Holmes, then chief technologist of the United States Geological Survey, and was continued by the Bureau of Mines after its establishment in 1910. The tests were confined to determinations of the loss in heating value of the coals and did not include a study of other deterioration; for example, in coking quality or the yield of by-products in coking. The Bureau of Yards and Docks of the Navy Department cooperated in tests of New River (W. Va.) coal, a variety largely used by the Navy. A preliminary report presenting a brief account of the early results of these tests has been published by the Bureau of Mines as Technical Paper 16.*

The detailed report is presented in this bulletin, which gives a full account of the tests and the analytical data covering a period of five years' storage. Data of somewhat similar experiments for shorter periods with gas coal from the Pittsburgh bed, with Pocahontas coal on the Isthmus of Panama, and with Sheridan, Wyo., subbituminous coal, which is used for railroad and other purposes in the West, are included.

The tests of New River coal, in cooperation with the Navy Department, were undertaken to determine the advantage to be gained by storing coal under water, and particularly under salt water. Small lots were used in order to make the tests of maximum severity, and parallel experiments were made with run-of-mine and crushed coal under one-fourth inch size. All of the small lots tested under the different conditions (except those tested near Key West, Fla.) were taken as representative portions from one large original lot.

The tests of Pocahontas coal, which, like the New River, is semi-bituminous, were undertaken chiefly to determine the effect of the

* Porter, H. C., and Ovitz, F. K., *Deterioration and spontaneous heating of coal in storage*, 1912, 13 pp.

tropical conditions in Panama. They were made on an outdoor pile of 100 tons of run-of-mine coal.

Pittsburgh gas coal, a high-volatile, bituminous type used at gas and by-product coke works, was exposed for test at Ann Arbor, Mich., in cooperation with the University of Michigan. The university agreed to make tests at successive intervals of storage to determine, in its illuminating-gas experiment station, the yields of gas and by-products from the coal. The coal, screened lump, was stored in about 4-ton lots out of doors in open bins.

Subbituminous coal, which is mined in Colorado, Wyoming, and other States, and is known also as "black lignite," is commonly supposed to deteriorate rapidly in storage, especially by "slacking" or crumbling of the lumps. The tests herein described were undertaken at Sheridan, Wyo., in cooperation with the Chicago, Burlington & Quincy Railroad Co., in order to determine the extent of this slacking and the accompanying loss of heat value. The tests were in open bins holding 4 to 12 tons.

GENERAL SUMMARY OF RESULTS.

The results of all these tests are to be taken as showing only the change in heating value and approximately, also, the degradation of lumps by weathering; as to any resultant deadening effect or decrease of original ease of burning, no examination or test was made.

In brief, it may be said, the tests show that the amount of deterioration of coal in heating value during storage has commonly been overestimated. Except for the subbituminous Wyoming coal, no loss was observed in outdoor weathering greater than 1.2 per cent in the first year, or 2.1 per cent in two years. The Wyoming coal suffered somewhat more loss, 2 to 3 per cent in the first year and as much as 5.5 per cent in three years. Details are given under the separate headings.

ACKNOWLEDGMENTS.

Acknowledgment is due to the commandants of the United States navy yards at Portsmouth, N. H., and Norfolk, Va., and of the United States naval station at Key West, Fla., for their cooperation and assistance in the tests on New River coal; to Prof. A. H. White and his assistants at the University of Michigan for carrying out the sampling in the tests of Pittsburgh coals; and to the officials of the Panama Railroad Co., and the Chicago, Burlington & Quincy Railroad Co. for cooperation in the tests of Pocahontas coal and those of Sheridan, Wyo., coal, respectively. The analytical laboratory at the Pittsburgh experiment station of the Bureau of Mines, under the direction of A. C. Fieldner, chemist, performed a large part of the analyses connected with the tests.

TESTS OF NEW RIVER COAL.

Briefly summarized, the tests show that submergence storage of New River coal effectively prevents deterioration of calorific value, and that storage of that coal in the open air causes only slight deterioration, about 1 per cent in one year's exposure and about 2 per cent in two years. After two years, the loss of heating value is continuous but very slow, reaching about 2.5 to 3 per cent in five years. With New River coal, therefore, the expense of underwater storage equipment is not justified except as an absolute preventive of fires from spontaneous combustion.

SOURCE AND PREPARATION OF THE COAL.

The coal used in the tests was from the Sun mine, working the Sewell bed in the New River district, Fayette County, W. Va., and was mined especially for this purpose, under the supervision of a Government mining engineer. An endeavor was made to obtain coal representing the commercial output of the mine.

Small quantities of coal, which in the majority of tests was finely crushed, were used for the express purpose of making the tests of maximum severity and in order to facilitate sampling.

On August 27, 1909, one lot of 3 to 4 tons of run-of-mine coal was collected at the mine and shipped in sacks to Washington, D. C. Three mine samples were taken at the same time from the faces from which the coal was mined, and were mailed at once in sealed cans to the laboratory for analysis. The test coal remained 16 days in the sacks, and then a representative portion, about 2 tons, was crushed to pass a one-fourth-inch screen, mixed well, and divided into small test portions for storing. Each of these portions was reduced by quartering and carefully sampled. Eighteen 50-pound lots of one-fourth-inch coal were placed in heavy wooden boxes for submergence under water. These boxes were lined with canvas and perforated with three-fourths-inch holes to facilitate displacement of the air during submergence. Eight lots of the crushed coal, 300 to 350 pounds each, were placed in barrels. From the balance of the original lot of coal, eight portions, also run-of-mine, were placed in barrels. Each barrel was sampled as thoroughly as possible by taking a number of well-distributed portions of lump and fine, then crushing and reducing these by quartering.

These test portions of coal as prepared in Washington were shipped by freight to the Portsmouth, N. H., and Norfolk, Va., navy yards, and to the experiment station of the Bureau of Mines at Pittsburgh, Pa. Table 1 gives the essential data as to number of portions stored.

On December 14, 1909, another lot of coal was collected from the same mine for storage near Key West, Fla. The mining, preparation,

and sampling of the coal were carried out as before, except that a longer time (47 days) elapsed between the mining of the coal and sampling, thereby allowing greater deterioration before the beginning of the tests. Wooden boxes used for the submergence tests near Key West were made of lumber that had been creosoted in order to prevent, if possible, destruction by the teredo. Some of these boxes remained intact, for the purposes of the test, during submergence for three and one-half years.

TABLE 1.—*Conditions of storage of New River, W. Va., coal.*

Location of test.	Days between mining and sampling.	Days between sampling and storing.	Size of coal.	Number of portions of—			
				Coal in boxes submerged under sea water.	Coal in barrels submerged under fresh water.	Coal exposed indoors.	Coal exposed outdoors.
Pittsburgh, Pa. . .	16	34	1-inch		1	1	2
			Run-of-mine		1	1	2
Portsmouth, N.H. .	16	85	1-inch	9		1	1
			Run-of-mine			1	1
Norfolk, Va.	16	41	1-inch	9		1	1
			Run-of-mine			1	1
Key West, Fla. . . .	47	27	1-inch	10		1	1
			Run-of-mine			1	1

STORING THE COAL.

At Pittsburgh, Pa., eight barrels of coal were placed in storage on September 30, 1909, as follows: One barrel of $\frac{1}{4}$ -inch coal and one of run-of-mine were filled with fresh water so as to submerge the coal; one barrel of each grade, dry, was placed indoors loosely covered, and two barrels of each grade were emptied in separate piles, fully exposed to the weather, on the roof of a building at the Bureau of Mines experiment station.

At Portsmouth, N. H., nine boxes and four barrels were placed in storage at the navy yard on November 20, 1909. These test portions had stood at the yard in their closed containers from September 20 to November 20. The nine boxes were placed in a larger box, which was sunk under the full salt water of the dry dock basin until it rested on the granite bottom. (See Pl. I.) The small test boxes were entirely submerged at all times, although the large box containing them was partly out of water at low tide. One barrel of $\frac{1}{4}$ -inch coal and one of run-of-mine, both with heads removed, were placed on a platform in the fire-engine house (see Pl. II, A), and one barrel of each grade was emptied in an open pile on the roof of a small shed, where it was fully exposed to the weather (see Pl. II, B).

At Norfolk, Va., nine boxes and four barrels were placed in storage at the navy yard on October 7, 1909. The boxes were submerged by chaining them to the pier at a depth insuring complete submer-



A. WEIGHTED BOX CONTAINING SMALLER BOXES OF NEW RIVER COAL SUBMERGED UNDER SALT WATER AT PORTSMOUTH, N. H.



B. DRY-DOCK BASIN, PORTSMOUTH, N. H., WHERE BOXES OF NEW RIVER COAL WERE SUBMERGED.



A. BARRELS OF NEW RIVER COAL EXPOSED INDOORS AT PORTSMOUTH, N. H.



B. NEW RIVER COAL EXPOSED TO WEATHER AT PORTSMOUTH, N. H.

gence at all tides (see Pl. III, A). The portions for open-air tests indoors were emptied from the barrels into open boxes on platforms, especially made for the purpose inside of one of the navy yard buildings (see Pl. III, C); and the outdoor open-air tests were similarly arranged outside one of the buildings (see Pl. III, B).

Near Key West, Fla., the test portions were placed in storage at Fort Jefferson, Dry Tortugas, on February 25, 1910, a special lot of coal having been collected for them nearly four months after the collection of the first lot. Ten creosoted boxes were submerged in the moat, one barrel of each grade was placed indoors in a casemate, and one of each set was put outside on the parapet, exposed to the weather. Two months later, on April 25, the outside test portions were emptied from the barrels into open piles. Early in January, 1911, 10 months after storing, the open-air test piles were scattered by a storm and lost. The two barrels that had been stored indoors were then emptied into outdoor piles to replace those lost.

At Portsmouth, N. H., the tests were under the immediate supervision of U. S. G. White, civil engineer, United States Navy, during the first few months of their progress, and of L. E. Gregory, civil engineer, United States Navy, thereafter. At Dry Tortugas, Fla., they were carried out by George C. Short, mate, United States Navy, under the direction of the commandant of the station. At Norfolk, Va., a representative of the Government fuel inspection service directed the starting of the tests and the subsequent sampling.

SAMPLING THE COAL.

Samples of each portion were taken periodically and the successive analyses were compared separately. During the first year, samples were taken every three months, during the second year every six months, and thereafter every year. The submerged boxes were sampled in rotation; that is, at three months box 1 was sampled, at six months box 2, and so on, in order to avoid repeated exposure of the coal. In addition, one particular box was repeatedly sampled in order to determine the effect, if any, of this periodical exposure on the deterioration. Thus at each sampling time two boxes were raised from the water, one was immediately replaced, the other was emptied. The wet coal in the latter was spread out in the open air for 24 hours, then sampled and replaced.

The submerged fine coal was sampled by the customary quartering method, the entire lot being spread out and reduced, by repeated quartering and rejection of alternate quarters, to a sample of convenient size for mailing, about 2 to 3 pounds. The barrel lots exposed to the air were not sampled in the same way because the coal would thus have been repeatedly turned over at each sampling time and unduly exposed as compared with usual storage conditions. Instead, samples were taken by selecting at random six or eight well-

distributed portions of 2 pounds each, then mixing and quartering them. From the run-of-mine coal, half of these selected portions were pieces broken from lumps, the balance being from the finer coal. Before quartering, all lumps were crushed to $\frac{1}{4}$ -inch size. In most cases duplicate samples were taken from each lot.

ANALYSIS OF THE COAL SAMPLES.

All coal samples taken during the first two years' progress of the tests were analyzed at the laboratory of the Pittsburgh experiment station, the calorimetric determinations being made throughout by the same man and with the same calorimeter. Each sample was analyzed for moisture, ash, and sulphur, and its calorific value determined. A number of composite samples were subjected to a complete ultimate analysis for carbon, hydrogen, oxygen, nitrogen, and sulphur. The methods used were those described in Technical Paper 8^a of the bureau.

"UNIT COAL" THE BASIS OF COMPARISON OF VALUES.

In studying the deterioration of coal the important practical problem is to determine the change in the dry organic substance. The incidental ingredients—sulphur, ash, and moisture—that accompany the coal substance affect the calorific value of a coal, but any changes that they may undergo have nothing to do with the alteration of the coal substance itself. For example, a coal wetted by exposure has its apparent calorific value reduced through the addition of water, whereas on the basis of dry material its fuel value may be unaltered. Furthermore, as to ash and sulphur, it is not practicable, by use of the sampling methods adopted in these tests, to obtain from the same lot of coal, even in $\frac{1}{4}$ -inch size, successive samples in which the aggregate percentages of these ingredients will always agree within 0.5 per cent. In other words, determination of calorific values of a stored coal might seem to show a deterioration of, say, 0.5 per cent, when in fact all of this change could be accounted for by the different moisture or ash content of the samples.

This investigation, therefore, endeavors to show merely the extent of deterioration in the actual coal substance, which in this paper is termed "unit coal." The calorific values as determined are all reduced to this basis—eliminating the effect of moisture, ash, and sulphur—by the following formula:

$$W = \frac{w - 2620S}{1.00 - [(M + A + \frac{1}{8}S + 0.04(A - \frac{1}{8}S))]}$$

in which W = calories on unit coal basis, w = determined calories, M = moisture, A = ash, and S = sulphur.

^aStanton, F. M., and Fieldner, A. C., Methods of analyzing coal and coke: Tech. Paper 8, Bureau of Mines, 1913, 42 pp.



A. DOCK UNDER WHICH BOXES OF NEW RIVER COAL WERE SUBMERGED AT NORFOLK, VA.

Black cross indicates spot under which boxes were chained to the piles.



B. OUTDOOR EXPOSURE OF NEW RIVER COAL AT NORFOLK, VA.



C. EXPOSURE OF NEW RIVER COAL UNDER SHELTER AT NORFOLK, VA.



As sulphur has a positive calorific value, its theoretical heat of combustion when in the form of iron pyrites, 2,620 calories, is subtracted from the determined calorific value. The expression below the line in the formula represents the percentage of "unit coal" in the sample, the moisture, ash, and sulphur being subtracted from 100 per cent. The expression, $A + \frac{1}{2}S + 0.04(A - \frac{1}{2}S)$ is based on the assumption that all the sulphur is present as FeS_2 (iron pyrites) and that therefore the ash content determined must be corrected for the decrease in weight due to burning FeS_2 to Fe_2O_3 —this decrease being five-eighths of the sulphur—and on the further assumption that the ash as determined is less than the true original ash by the amount of combined water driven out of clay and shale, roughly 4 per cent of the pyrite-free ash. These assumptions are based on a chemical examination of the shale and sulphur partings of the coal bed in the mine from which this coal was obtained.

WEATHER CONDITIONS.

Weather conditions varied at the different storage points, the tests being more severe at Key West and Norfolk than at Portsmouth and Pittsburgh. Table 2 shows average temperature conditions in air and water at the different points throughout the first two years of the tests.

TABLE 2.—*Monthly temperature averages (° F.) of air and water.*

[P.—Portsmouth, N. H.; Pgh.—Pittsburgh; N.—Norfolk; K. W.—Key West (Dry Tortugas).]

Month.	Temperature of air.								Temperature of water.			
	Average.				Maximum.				Average.			
	P.	Pgh.	N.	K.W.	P.	Pgh.	N.	K.W.	P.	Pgh. ^a	N.	K.W. ^b
1909.												
October.....	51.1	49.6	58.4	77.4	85	80	79	88	85			78
November.....	43.5	50.4	55.0	73.4	72	72	76	82	90	57		77
December.....	28.6	29.8	38.8	69.2	52	65	64	81	60	42		71
1910.												
January.....	28.5	30.8	41.6	67.1	51	51	69	80	55	43		66
February.....	25.4	28.6	42.2	69.0	54	55	76	81	55	41		67
March.....	39.4	48.8	55.0	71.3	74	83	90	81	39	60	51	69
April.....	48.6	52.8	60.0	74.2	68	83	87	83	47	65	60	72
May.....	55.8	57.8	65.2	78.4	78	83	90	87	50	70	66	77
June.....	65.4	67.0	72.0	81.9	90	90	92	89	57	75	74	81
July.....	74.7	74.9	78.4	82.8	94	91	93	90	61	75	81	82
August.....	69.4	72.8	76.4	83.1	90	90	90	92	60	73	81	83
September.....	62.7	67.6	72.4	81.4	83	86	92	90	59	70	77	82
October.....	54.6	57.6	63.8	77.8	87	84	86	87	53	70	70	79
November.....	40.9	37.1	45.4	70.4	59	63	71	79	46	65	49	73
December.....	26.2	27.0	37.2	65.3	51	55	64	78	37	60	39	70
1911.												
January.....	28.3	35.2	45.0	70.3	56	60	72	80	35	60	42	66
February.....	24.0	25.3	43.8	71.5	46	64	71	82	33	60	45	67
March.....	32.8	37.1	47.0	73.5	55	66	77	84	36	65	47	69
April.....	44.7	48.1	54.6	77.1	82	78	80	85	40	65	56	75
May.....	61.9	68.0	69.2	77.6	99	93	94	86	49	65	68	77
June.....	65.1	71.0	75.6	81.8	93	94	98	89	56	70	77	80
July.....	76.6	75.4	79.2	82.2	106	109	97	90	60	75	83	83
August.....	69.2	73.8	78.8	82.0	93	97	94	90	58	75	81	83
September.....	62.0	68.0	74.4	82.5	87	86	88	90	56	70	79	82
October.....	52.0	54.2	62.6	81.8	73	77	89	89	50	70	68	81

^a Estimated. The barrels of coal were transferred in December, 1909, from a warm room to a cooler one.

^b Estimated, except in January, April, and July, 1911, when observations were made on single days.

RESULTS OF STORAGE TESTS.

Results of storage tests are shown in Tables 3 to 16 following:

TABLE 3.—Storage tests of New River, W. Va., coal, $\frac{1}{4}$ -inch crushed, submerged under sea water, at Portsmouth, N. H., in 50-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Mine sample.....	Aug. 27, 1909	3	P. ct. 2.46	P. ct. 2.50	P. ct. 0.66	8,438	15,188	8,773	15,792	P. ct.
As stored:										
Box 1.....	Sept. 12, 1909	2	1.98	4.03	.74	8,361	15,659	8,747	15,745	
2.....	do.	2	1.84	4.39	.76	8,343	15,018	8,763	15,774	
3.....	do.	2	2.07	5.99	.98	8,183	14,739	8,780	15,750	
4.....	do.	2	2.14	5.00	.78	8,288	14,918	8,761	15,770	
5.....	do.	2	2.18	5.14	.92	8,277	14,899	8,769	15,784	
6.....	do.	2	1.92	4.27	.76	8,357	15,043	8,764	15,775	
7.....	do.	2	1.39	4.88	.76	8,298	14,936	8,780	15,767	
8.....	do.	2	1.79	5.89	.94	8,196	14,752	8,755	15,759	
9.....	do.	2	1.72	4.75	.88	8,298	14,837	8,752	15,754	
After 4 months:										
Box 1.....	Jan. 3, 1910	1	19.77	7.60	2.28	7,986	14,375	8,730	15,714	0.2
After 6 months:										
Box 2.....	Apr. 15, 1910	1	14.83	6.21	.95	8,113	14,603	8,700	15,660	.7
9.....	do.	2	5.65	5.84	.91	8,167	14,701	8,715	15,687	.4
After 9 months:										
Box 3.....	July 19, 1910	1	16.68	6.27	.90	8,151	14,672	8,744	15,739	.1
9.....	do.	2	14.74	4.76	.80	8,300	14,939	8,751	15,752	
After 1 year:										
Box 4.....	Oct. 11, 1910	2	17.42	4.33	.75	8,347	15,025	8,759	15,765	
9.....	do.	2	13.71	3.94	.71	8,386	15,112	8,773	15,791	.2
After 1 $\frac{1}{2}$ years:										
Box 5.....	Apr. 7, 1911	2	19.91	7.08	1.28	8,047	14,485	8,724	15,703	.5
9.....	do.	2	17.57	6.52	1.01	8,095	14,571	8,711	15,680	.5
After 2 years:										
Box 6.....	Oct. 10, 1911	1	16.91	5.54	.83	8,201	14,762	8,724	15,703	.5
9.....	do.	2	18.72	6.60	.94	8,111	14,599	8,733	15,719	.2
After 3 years:										
Box 7.....	Oct. 17, 1912	2	18.39	5.97	1.01	8,134	14,641	8,698	15,614	1.0
9.....	do.	1	22.22	7.37	1.14	7,958	14,324	8,648	15,566	1.2
After 4 years:										
Box 8.....	Oct. 6, 1913	2	17.72	7.40	1.19	8,019	14,434	8,718	15,692	.4
9.....	do.	2	21.58	8.34	1.17	7,909	14,236	8,699	15,640	.6
After 5 years:										
Box 8.....	Oct. 17, 1914	2	17.93	8.49	1.30	7,922	14,259	8,723	15,701	.4

= Gain.

TABLE 4.—Storage tests of New River, W. Va., coal, exposed indoors at Portsmouth, N. H., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run of mine coal:										
As stored.....	Sept. 12, 1909	2	P. ct. 1.75	P. ct. 4.45	P. ct. 0.69	8,332	14,998	8,574	15,757	P. ct.
After 3 months.....	Jan. 3, 1910	1	.67	4.82	1.33	8,282	14,908	8,753	15,756	0.0
1 year.....	Oct. 11, 1910	2	1.19	2.94	.46	8,499	15,254	8,748	15,746	.1
1 $\frac{1}{2}$ years.....	Apr. 7, 1911	2	.70	6.28	1.28	8,115	14,606	8,713	15,683	.5
2 years.....	Oct. 10, 1911	2	1.01	3.98	.74	8,350	15,029	8,728	15,710	.3
3 years.....	Oct. 11, 1912	2	.98	5.34	1.09	8,168	14,703	8,675	15,615	.9
4 years.....	Oct. 8, 1913	2	.98	6.25	1.33	8,085	14,553	8,681	15,625	.8
5 years.....	Oct. 17, 1914	2	1.15	5.73	.94	8,124	14,623	8,663	15,593	1.0
$\frac{1}{4}$ -inch crushed coal:										
As stored.....	Sept. 12, 1909	2	1.79	6.27	.84	8,185	14,733	8,779	15,802	
After 3 months.....	Jan. 3, 1910	1	.92	7.82	.94	7,966	14,339	8,724	15,703	.6
1 year.....	Oct. 11, 1910	2	1.22	5.64	.70	8,213	14,782	8,742	15,736	.4
1 $\frac{1}{2}$ years.....	Apr. 7, 1911	2	.79	6.68	.86	8,096	14,572	8,721	15,679	.7
2 years.....	Oct. 10, 1911	2	1.23	7.14	.97	8,034	14,480	8,703	15,666	.9
3 years.....	Oct. 11, 1912	2	1.02	6.42	.94	8,067	14,521	8,667	15,601	1.3
4 years.....	Oct. 8, 1913	2	1.04	6.93	1.01	7,995	14,391	8,641	15,554	1.6
5 years.....	Oct. 17, 1914	2	1.19	6.84	1.02	8,012	14,422	8,652	15,573	1.4

TABLE 5.—Storage tests of New River, W. Va., coal exposed to weather at Portsmouth, N. H., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:										
As stored.....	Sept. 12, 1909	2	P. ct.	P. ct.	P. ct.					P. ct.
After 3 months.....	Jan. 3, 1910	1	1.81	5.34	0.70	8,271	14,886	8,776	15,796	
6 months.....	Apr. 15, 1910	1	3.22	4.89	1.23	8,282	14,908	8,757	15,763	0.2
9 months.....	July 19, 1910	1	1.44	2.67	.53	8,486	15,375	8,742	15,736	.4
1 year.....	Oct. 11, 1910	2	1.28	5.02	.64	8,265	14,876	8,738	15,725	.4
1½ years.....	Apr. 7, 1911	2	2.32	2.20	.34	8,551	15,392	8,760	15,768	.2
2 years.....	Oct. 10, 1911	2	2.43	2.41	.53	8,507	15,313	8,738	15,728	.4
2½ years.....	Oct. 11, 1912	2	2.33	3.61	.38	8,378	15,060	8,716	15,689	.7
3 years.....	Oct. 7, 1913	2	4.80	3.67	.71	8,333	14,909	8,681	15,625	1.1
4 years.....	Oct. 7, 1913	2	3.07	4.18	.62	8,283	14,928	8,684	15,632	1.0
5 years.....	Oct. 17, 1914	2	3.45	3.53	.54	8,318	14,973	8,648	15,567	1.4
½-inch crushed coal:										
As stored.....	Sept. 12, 1909	2	2.01	6.68	1.02	8,116	14,009	8,748	15,747	
After 3 months.....	Jan. 3, 1910	2	3.10	7.29	1.27	8,028	14,451	8,720	15,695	.3
6 months.....	Apr. 15, 1910	1	2.00	6.48	1.03	8,111	14,000	8,726	15,707	.3
9 months.....	July 19, 1910	2	1.93	6.70	1.07	8,085	14,553	8,719	15,694	.3
1 year.....	Oct. 11, 1910	2	3.75	6.94	.94	8,661	14,491	8,701	15,661	.5
1½ years.....	Apr. 7, 1911	2	10.50	6.87	1.00	8,069	14,506	8,685	15,633	.7
2 years.....	Oct. 10, 1911	2	17.42	7.04	.75	8,016	14,429	8,668	15,603	.9
2½ years.....	Oct. 11, 1912	2	12.66	6.91	.92	7,985	14,360	8,637	15,547	1.3
3 years.....	Oct. 7, 1913	2	16.23	6.90	.72	8,004	14,407	8,641	15,553	1.2
5 years.....	Oct. 17, 1914	2	13.09	7.92	.77	7,908	14,234	8,637	15,547	1.3

TABLE 6.—Storage tests of New River, W. Va., coal, ½-inch crushed, submerged under sea water at Norfolk, Va., in 50-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
As stored:										
Box 10.....	Sept. 12, 1909	2	P. ct.	P. ct.	P. ct.					P. ct.
11.....	do.	2	2.00	5.20	0.96	8,272	14,801	8,772	15,790	
12.....	do.	2	2.01	6.45	.99	8,130	14,635	8,742	15,736	
13.....	do.	2	1.99	5.13	.82	8,260	14,867	8,744	15,789	
14.....	do.	2	1.89	5.42	.93	8,237	14,827	8,754	15,767	
15.....	do.	2	1.64	5.44	1.03	8,264	14,821	8,764	15,757	
16.....	do.	2	1.96	5.22	.81	8,269	14,804	8,764	15,775	
17.....	do.	2	1.64	5.12	.70	8,262	14,872	8,745	15,741	
After 3 months:										
Box 10.....	Jan. 11, 1910	2	17.54	5.82	1.05	8,191	14,744	8,747	15,745	0.3
After 6 months:										
Box 10.....	Apr. 7, 1910	2	10.47	5.72	.99	8,190	14,743	8,733	15,720	.4
11.....	do.	1	15.52	6.34	1.11	8,140	14,653	8,744	15,739	.0
After 10 months:										
Box 10.....	July 13, 1910	2	18.06	5.35	.92	8,248	14,846	8,757	15,763	.2
12.....	do.	1	16.62	4.56	.74	8,310	14,968	8,744	15,739	.0
After 1 year:										
Box 10.....	Oct. 4, 1910	2	14.94	5.72	.95	8,198	14,748	8,737	15,725	.4
12.....	do.	1	17.35	4.34	.82	8,345	15,021	8,763	15,773	.1
After 1½ years:										
Box 10.....	Apr. 4, 1911	2	20.68	6.67	1.13	8,105	14,588	8,738	15,729	.4
14.....	do.	1	19.71	6.69	1.28	8,114	14,605	8,756	15,761	.0
After 2 years:										
Box 10.....	Oct. 4, 1911	2	18.92	8.01	1.12	7,968	14,344	8,719	15,695	.6
15.....	do.	1	20.47	6.55	.93	8,108	14,593	8,725	15,705	.4
After 3 years:										
Box 10.....	Oct. 15, 1912	2	24.76	11.03	1.27	7,641	13,753	8,663	15,594	1.2
16.....	do.	1	16.46	6.92	.97	8,047	14,485	8,696	15,653	.6
After 4 years:										
Box 17.....	Oct. 10, 1913	1	20.98	6.77	1.08	8,069	14,524	8,708	15,674	.4
After 5 years:										
Box 17.....	Oct. 8, 1914	1	24.62	7.91	1.15	7,957	14,323	8,701	15,662	.5

* Gain.

TABLE 7.—Storage tests of New River, W. Va., coal exposed indoors at Norfolk, Va., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:			<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>					<i>P. ct.</i>
As stored.....	Sept. 12, 1909	2	1.61	4.14	0.77	8,348	15,037	8,743	15,737	
After 3 months.....	Jan. 11, 1910	1	1.37	4.39	.65	8,349	15,029	8,764	15,775	
6 months.....	Apr. 7, 1910	2	1.14	3.93	.65	8,367	15,061	8,739	15,729	
9 months.....	July 12, 1910	2	1.43	5.60	.84	8,302	14,782	8,732	15,718	
1 year.....	Oct. 4, 1910	2	1.47	5.10	.71	8,246	14,843	8,725	15,705	
1½ years.....	Apr. 4, 1911	2	1.49	4.70	.73	8,282	14,658	8,727	15,709	
2 years.....	Oct. 4, 1911	2	1.33	5.35	.75	8,221	14,798	8,724	15,703	
3 years.....	Oct. 15, 1912	2	4.67	5.18	.88	8,207	14,773	8,686	15,653	
4 years.....	Oct. 11, 1913	2	1.81	4.20	.74	8,287	14,917	8,684	15,631	
5 years.....	Oct. 8, 1914	2	1.83	6.01	.87	8,110	14,598	8,673	15,612	
½-inch crushed coal—										
As stored.....	Sept. 12, 1909	2	1.78	5.67	.85	8,216	14,787	8,751	15,752	
After 3 months.....	Jan. 11, 1910	1	1.52	5.88	.79	8,188	14,739	8,742	15,736	
6 months.....	Apr. 7, 1910	2	1.28	6.02	.73	8,157	14,683	8,718	15,696	
9 months.....	July 12, 1910	2	1.54	5.71	.69	8,189	14,740	8,725	15,705	
1 year.....	Oct. 4, 1910	2	1.70	5.68	.64	8,208	14,774	8,736	15,724	
1½ years.....	Apr. 4, 1911	2	1.60	5.88	.69	8,180	14,687	8,708	15,674	
2 years.....	Oct. 4, 1911	2	1.54	6.04	.74	8,153	14,674	8,718	15,692	
3 years.....	Oct. 15, 1912	2	4.54	6.66	.88	8,041	14,473	8,661	15,590	
4 years.....	Oct. 11, 1913	2	1.73	7.14	.82	7,988	14,375	8,647	15,565	
5 years.....	Oct. 8, 1914	2	2.06	6.84	.82	8,011	14,420	8,646	15,562	

* Gain.

TABLE 8.—Storage tests of New River, W. Va., coal exposed to weather at Norfolk, Va., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:			<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>					<i>P. ct.</i>
As stored.....	Sept. 12, 1909	2	1.74	3.63	0.58	8,397	15,115	8,743	15,738	
After 3 months.....	Jan. 11, 1910	2	3.04	4.20	.74	8,345	15,021	8,745	15,741	
6 months.....	Apr. 7, 1910	2	3.30	4.76	.65	8,282	14,909	8,732	15,717	
9 months.....	July 12, 1910	2	2.06	4.41	.59	8,336	15,004	8,752	15,754	
1 year.....	Oct. 4, 1910	2	2.06	5.19	.68	8,209	14,817	8,720	15,695	
1½ years.....	Apr. 4, 1911	2	9.90	5.05	.52	8,258	14,865	8,729	15,711	
2 years.....	Oct. 4, 1911	2	3.44	5.77	.65	8,187	14,683	8,695	15,650	
3 years.....	Oct. 15, 1912	2	6.80	5.41	.64	8,158	14,684	8,680	15,587	
4 years.....	Oct. 11, 1913	2	6.25	4.60	.61	8,209	14,776	8,637	15,646	
5 years.....	Oct. 8, 1914	2	6.59	6.19	.63	8,075	14,535	8,646	15,563	
½-inch crushed coal:										
As stored.....	Sept. 12, 1909	2	2.04	7.97	1.00	7,977	14,358	8,725	15,704	
After 3 months.....	Jan. 11, 1910	2	4.26	8.61	1.11	7,907	14,233	8,715	15,686	
6 months.....	Apr. 7, 1910	2	5.83	9.98	1.04	7,784	14,010	8,708	15,675	
9 months.....	July 12, 1910	2	1.48	8.45	.95	7,910	14,227	8,698	15,657	
1 year.....	Oct. 4, 1910	2	1.80	8.89	.93	7,843	14,118	8,666	15,598	
1½ years.....	Apr. 4, 1911	2	14.38	9.52	.91	7,785	14,013	8,664	15,594	
2 years.....	Oct. 4, 1911	2	5.38	10.84	.90	7,631	13,784	8,624	15,522	
3 years.....	Oct. 15, 1912	2	13.48	10.83	.85	7,606	13,691	8,594	15,469	
4 years.....	Oct. 11, 1913	2	11.92	12.49	.73	7,437	13,386	8,567	15,420	
5 years.....	Oct. 8, 1914	2	15.73	10.62	.71	7,647	13,764	8,614	15,505	

* Gain.

TABLE 9.—Storage tests of New River, W. Va., coal submerged in fresh water at Pittsburgh, Pa., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:										
As stored.....	Sept. 12, 1909	2	P. ct.	P. ct.	P. ct.	8,299	14,885	8,756	15,761	P. ct.
After 3 months.....	Jan. 22, 1910	1	8.11	7.17	.95	8,072	14,530	8,748	15,746	0.1
6 months.....	Apr. 8, 1910	1	10.72	5.50	.76	8,230	14,814	8,752	15,754	.0
9 months.....	July 8, 1910	2	7.83	5.96	1.23	8,197	14,754	8,770	15,786	.2
1 year.....	Oct. 7, 1910	2	8.97	6.11	1.15	8,175	14,715	8,762	15,771	.1
1½ years.....	Apr. 13, 1911	2	9.65	5.18	1.04	8,256	14,860	8,751	15,751	.1
2 years.....	Oct. 9, 1911	2	12.09	7.36	.98	8,075	14,534	8,769	15,784	.1
3 years.....	Oct. 4, 1912	2	7.32	5.69	1.24	8,170	14,706	8,707	15,672	.6
4 years.....	Oct. 7, 1913	2	8.96	5.89	1.01	8,184	14,730	8,743	15,737	.2
5 years.....	Oct. 15, 1914	2	10.83	9.30	2.41	7,844	14,119	8,749	15,747	.1
½-inch crushed coal:										
As stored.....	Sept. 12, 1909	2	1.58	4.83	.64	8,299	14,939	8,752	15,754	
After 3 months.....	Jan. 22, 1910	1	13.04	5.28	.68	8,263	14,873	8,762	15,772	.1
6 months.....	Apr. 8, 1910	1	22.62	5.47	.68	8,234	14,804	8,740	15,732	.0
9 months.....	July 8, 1910	2	19.64	5.18	.64	8,268	14,882	8,756	15,761	.1
1 year.....	Oct. 7, 1910	2	20.04	5.35	.61	8,245	14,841	8,747	15,744	.1
1½ years.....	Apr. 13, 1911	2	23.44	5.27	.64	8,248	14,847	8,743	15,738	.1
2 years.....	Oct. 9, 1911	2	19.78	5.69	.67	8,229	14,795	8,754	15,767	.0
3 years.....	Oct. 4, 1912	2	18.82	5.62	.75	8,163	14,693	8,663	15,639	.7
4 years.....	Oct. 7, 1913	1	20.26	5.84	.74	8,192	14,746	8,741	15,734	.1
5 years.....	Oct. 15, 1914	2	21.76	6.23	.80	8,160	14,687	8,745	15,741	.1

* Gain.

TABLE 10.—Storage tests of New River, W. Va., coal exposed indoors at Pittsburgh, Pa., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:										
As stored.....	Sept. 12, 1909	2	P. ct.	P. ct.	P. ct.	8,271	14,887	8,753	15,756	
After 3 months.....	Jan. 22, 1910	1	.80	2.91	.72	8,383	15,099	8,758	15,764	0.1
6 months.....	Apr. 8, 1910	2	.92	4.31	.89	8,301	14,942	8,715	15,687	.4
9 months.....	July 8, 1910	2	1.19	4.10	.71	8,360	15,047	8,750	15,749	.0
1 year.....	Oct. 7, 1910	2	1.43	4.63	.66	8,280	14,903	8,741	15,733	.1
1½ years.....	Apr. 13, 1911	2	.87	6.71	1.52	8,072	14,530	8,717	15,690	.4
2 years.....	Oct. 9, 1911	2	1.62	5.54	.74	8,216	14,799	8,737	15,727	.2
3 years.....	Oct. 4, 1912	2	1.80	2.60	.54	8,323	14,962	8,659	15,586	1.1
4 years.....	Oct. 7, 1913	2	1.83	4.40	.92	8,253	14,856	8,671	15,608	.9
5 years.....	Oct. 15, 1914	2	2.22	5.01	1.07	8,219	14,794	8,698	15,556	.6
½-inch crushed coal:										
As stored.....	Sept. 12, 1909	2	1.68	6.08	1.04	8,180	14,740	8,769	15,787	
After 3 months.....	Jan. 22, 1910	1	2.73	6.69	1.19	8,103	14,585	8,740	15,732	.3
6 months.....	Apr. 8, 1910	2	.99	6.52	1.12	8,109	14,596	8,727	15,709	.5
9 months.....	July 8, 1910	2	1.39	6.12	1.01	8,152	14,673	8,733	15,719	.4
1 year.....	Oct. 7, 1910	2	1.48	6.08	.93	8,091	14,564	8,719	15,693	.6
1½ years.....	Apr. 13, 1911	2	1.08	6.64	1.11	8,068	14,523	8,695	15,650	.9
2 years.....	Oct. 9, 1911	2	4.14	8.83	1.03	7,898	14,198	8,714	15,685	.6
3 years.....	Oct. 4, 1912	2	2.99	10.27	1.20	7,678	13,820	8,626	15,527	1.6
4 years.....	Oct. 7, 1913	1	2.50	8.55	1.16	7,834	14,101	8,629	15,532	1.6
5 years.....	Oct. 15, 1914	2	2.81	9.26	1.20	7,763	13,974	8,620	15,515	1.7

TABLE 11.—*Storage tests of New River, W. Va., coal exposed to weather at Pittsburgh, Pa., in 350-pound portions.*

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:			<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>					<i>P. ct.</i>
As stored.....	Sept. 12, 1909	4	1.14	4.04	0.59	8,370	15,066	8,752	15,754	
After 3 months.....	Jan. 24, 1910	2	4.39	7.69	1.46	8,007	14,412	8,740	15,732	0.1
6 months.....	Apr. 22, 1910	2	2.35	6.21	.71	8,126	15,627	8,704	15,667	.6
9 months.....	July 12, 1910	2	1.49	4.24	.62	8,348	15,026	8,740	15,745	.0
1 year.....	Oct. 14, 1910	2	1.86	5.01	.70	8,245	14,841	8,716	15,690	.4
1½ years.....	Apr. 20, 1911	4	3.78	6.08	.89	8,136	14,646	8,708	15,674	.5
2 years.....	Oct. 12, 1911	2	4.85	8.28	.78	7,923	14,261	8,690	15,642	.7
3 years.....	Oct. 7, 1912	2	2.32	5.85	.89	8,096	14,573	8,642	15,555	1.3
4 years.....	Oct. 8, 1913	2	3.09	5.20	.61	8,178	14,721	8,661	15,589	1.0
5 years.....	Oct. 13, 1914	2	4.26	8.43	.62	7,877	14,179	8,650	15,570	1.2
¼-inch crushed coal:										
As stored.....	Sept. 12, 1909	4	1.29	7.45	.91	8,052	14,494	8,752	15,754	
After 3 months.....	Jan. 24, 1910	2	14.02	8.75	1.22	7,904	14,227	8,726	15,707	.3
6 months.....	Apr. 22, 1910	2	1.71	7.23	1.15	8,014	14,426	8,695	15,651	.7
9 months.....	July 12, 1910	2	1.22	7.79	1.05	7,971	14,348	8,701	15,662	.6
1 year.....	Oct. 14, 1910	2	1.99	8.33	1.06	7,918	14,253	8,697	15,655	.6
1½ years.....	Apr. 20, 1911	4	6.67	9.49	1.27	7,798	14,036	8,685	15,632	.8
2 years.....	Oct. 12, 1911	2	10.88	9.31	1.18	7,792	14,026	8,659	15,586	1.1
3 years.....	Oct. 7, 1912	2	2.92	8.75	.98	7,805	14,049	8,612	15,502	1.6
4 years.....	Oct. 8, 1913	2	6.36	8.60	.82	7,827	14,088	8,616	15,509	1.6
5 years.....	Oct. 13, 1914	1	11.67	7.19	.62	7,975	14,355	8,637	15,547	1.3

TABLE 12.—*Storage tests of New River, W. Va., coal, ¼-inch crushed, submerged in sea water at Key West (Dry Tortugas), Fla., in 50-pound portions.*

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Mine sample.....	Dec. 15, 1909	2	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>					<i>P. ct.</i>
As stored:			3.20	7.26	1.99	8,053	14,495	8,761	15,770	
Box 35.....	Jan. 29, 1910	2	.79	7.09	2.13	8,045	14,480	8,742	15,735	
36.....	do.....	2	1.02	6.55	.81	8,108	14,593	8,722	15,700	
37.....	do.....	2	.89	4.10	.82	8,362	15,052	8,755	15,759	
38.....	do.....	2	1.06	3.24	.81	8,434	15,180	8,747	15,745	
39.....	do.....	2	1.02	3.96	.74	8,388	15,099	8,766	15,779	
40.....	do.....	2	.92	5.21	.80	8,253	14,856	8,745	15,741	
41.....	do.....	2	.96	4.87	.89	8,282	14,908	8,746	15,743	
42.....	do.....	2	.97	4.41	.83	8,335	15,003	8,757	15,763	
43.....	do.....	2	.97	5.92	.95	8,180	14,723	8,739	15,730	
44.....	do.....	2	1.02	5.73	.90	8,193	14,747	8,734	15,721	
After 3 months:										
Box 35.....	Apr. 26, 1910	2	7.74	6.23	1.32	8,126	14,626	8,723	15,702	0.2
36.....	do.....	2	9.59	6.31	.82	8,116	14,638	8,706	15,671	.2
After 6 months:										
Box 35.....	July 14, 1910	2	17.84	6.70	1.46	8,088	14,559	8,731	15,716	.1
37.....	do.....	2	11.93	7.33	.88	8,027	14,449	8,712	15,681	.5
After 8 months:										
Box 35.....	Oct. 4, 1910	1	18.89	7.53	1.66	8,034	14,461	8,762	15,772	a. 2
37.....	do.....	2	19.59	5.91	.91	8,163	14,694	8,720	15,695	.4
After 1 year:										
Box 35.....	Jan. 10, 1911	2	16.60	7.38	1.59	8,042	14,493	8,752	15,754	a. 1
38.....	do.....	2	12.68	4.54	.88	8,281	14,906	8,714	15,685	.4
After 1½ years:										
Box 35.....	May 1, 1911	2	18.54	7.76	1.99	7,990	14,383	8,744	15,739	.0
39.....	do.....	2	14.97	4.48	.89	8,305	14,949	8,733	15,720	.4
After 1½ years:										
Box 35.....	July 10, 1911	2	14.02	8.11	2.13	7,932	14,277	8,718	15,693	.3
40.....	do.....	2	18.78	6.68	.97	8,106	14,590	8,736	15,724	.1
After 2 years:										
Box 35.....	Jan. 10, 1912	2	12.64	8.72	2.57	7,882	14,187	8,734	15,721	.1
41.....	do.....	2	19.31	5.20	.86	8,275	14,895	8,770	15,786	a. 3
After 2½ years:										
Box 35.....	July 8, 1912	2	13.69	8.50	2.73	7,872	14,169	8,706	15,671	.4
42.....	do.....	1	18.88	5.50	1.17	8,195	14,751	8,722	15,700	.4
After 3 years:										
Box 35.....	Feb. 1, 1913	2	17.85	7.88	2.38	7,895	14,210	8,661	15,599	.9
42.....	do.....	1	20.89	8.31	2.46	7,865	14,157	8,674	15,613	1.0
After 3½ years:										
Box 43.....	Oct. 15, 1913	1	21.04	5.74	1.22	8,168	14,702	8,718	15,692	.2
44.....	do.....	2	10.00	7.20	1.24	8,011	14,420	8,692	15,645	.5

a Gain.

TABLE 13.—Storage tests of New River, W. Va., coal exposed indoors at Key West (Dry Tortugas), Fla., in 350-pound portions.*

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:			P. ct.	P. ct.	P. ct.					P. ct.
As stored.....	Jan. 29, 1910	2	0.91	8.29	2.15	7,935	14,283	8,741	15,734	
After 3 months.....	Apr. 26, 1910	2	1.19	11.18	3.11	7,642	13,756	8,733	15,790	0.1
6 months.....	July 14, 1910	1	1.26	10.27	3.54	7,697	13,855	8,712	15,682	.3
8 months.....	Oct. 4, 1910	2	1.14	11.67	5.45	7,539	13,570	8,731	15,715	.1
1 year.....	Jan. 10, 1911	1	1.83	10.79	4.30	7,628	13,730	8,710	15,678	.4
1-inch crushed coal:										
As stored.....	Jan. 29, 1910	2	.75	7.15	2.33	8,068	14,523	8,777	15,799	
After 3 months.....	Apr. 26, 1910	2	2.99	7.40	2.15	8,051	14,492	8,778	15,800	.0
6 months.....	July 14, 1910	1	1.80	7.40	1.74	7,984	14,371	8,694	15,649	1.0
8 months.....	Oct. 4, 1910	2	1.63	7.83	1.84	7,652	14,314	8,703	15,666	.8
1 year.....	Jan. 10, 1911	1	2.12	7.84	1.94	7,922	15,062	8,676	15,617	1.2

* These test portions were transferred at the end of 1 year from indoor storage to outdoor.

TABLE 14.—Storage tests of New River, W. Va., coal exposed to weather at Key West (Dry Tortugas), Fla., in 350-pound portions.

	Date sampled.	Number of samples averaged.	Moisture.	Analysis on dry basis.				Heating value on "unit coal" basis.		
				Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.	Loss in B. t. u.
Run-of-mine coal:			P. ct.	P. ct.	P. ct.					P. ct.
Lot A—As stored.....	Jan. 29, 1910	2	0.80	11.03	3.38	7,668	13,803	8,754	15,757	
Lot B—As stored.....	do.....	2	.91	8.29	2.15	7,935	14,283	8,741	15,734	
Lot A—										
After 3 months.....	Apr. 26, 1910	2	1.30	8.85	2.60	7,843	14,118	8,706	15,671	0.5
6 months.....	July 14, 1910	1	2.06	9.29	1.74	7,822	14,080	8,706	15,671	.5
8 months.....	Oct. 4, 1910	2	1.73	4.94	1.62	8,165	14,696	8,703	15,666	.6
1 year.....	Jan. 10, 1911	1	1.91	5.91	2.67	8,137	14,647	8,723	15,700	.4
Lot B—										
After 1½ years.....	May 1, 1911	2	1.61	9.62	3.18	7,742	13,936	8,688	15,638	.6
1½ years.....	July 10, 1911	1	1.74	7.49	2.07	7,943	14,297	8,670	15,606	.8
2 years.....	Jan. 10, 1912	1	2.06	7.37	1.33	7,937	14,287	8,632	15,638	1.2
2½ years.....	July 8, 1912	1	1.08	7.20	1.64	7,916	14,249	8,598	15,476	1.6
3 years.....	Feb. 1, 1913	1	1.79	9.99	1.70	7,679	13,822	8,513	15,323	2.6
1-inch crushed coal:										
Lot A—As stored.....	Jan. 29, 1910	2	1.06	4.00	.73	8,362	15,062	8,745	15,740	
Lot B—As stored.....	do.....	2	.75	7.15	2.33	8,068	14,523	8,777	15,799	
Lot A—										
After 3 months.....	Apr. 26, 1910	2	2.28	5.57	1.11	8,189	14,741	8,721	15,666	.3
6 months.....	July 14, 1910	1	1.27	5.59	1.14	8,172	14,710	8,706	15,671	.4
8 months.....	Oct. 4, 1910	2	1.80	5.70	1.05	8,150	14,670	8,690	15,642	.6
1 year.....	Jan. 10, 1911	1	3.16	7.97	1.17	7,933	14,279	8,680	15,624	.7
Lot B—										
After 1½ years.....	May 1, 1911	2	1.36	6.69	1.65	8,038	14,468	8,681	15,625	1.1
1½ years.....	July 10, 1911	1	1.38	6.58	1.83	7,983	14,369	8,618	15,512	1.8
2 years.....	Jan. 10, 1912	1	4.27	6.87	1.65	7,941	14,294	8,592	15,466	2.1
2½ years.....	July 8, 1912	1	1.19	6.71	1.90	7,907	14,233	8,547	15,385	2.6
3 years.....	Feb. 1, 1913	1	12.96	7.88	1.67	7,817	14,071	8,557	15,403	2.5

TABLE 15.—General summary of tests of New River coal.

Condition of test.	Percentage loss in heat value during—								
	3 months.	6 months.	9 months.	1 year.	1½ years.	2 years.	3 years.	4 years.	5 years.
Submerged:									
Portsmouth, N. H.....	0.3	0.5	0.1	0.0	0.5	0.3	1.1	0.5	0.4
Norfolk, Va.....	.2	.2	.1	.2	.2	.5	.9	.4	.5
Key West, Fla.....	.2	.3	.1	.2	.2	.0	.9	.4	
Pittsburgh, Pa.—									
Run-of-mine coal.....	.1	.0	.0	.0	.1	.0	.6	.2	.1
½-inch crushed coal.....	.0	.1	.0	.1	.1	.0	.7	.1	.1
Average.....	.16	.55	.06	.10	.55	.16	.84	.33	.25
Exposed indoors:									
Run-of-mine coal—									
Portsmouth, N. H.....	.0			.1	.5	.3	.9	.8	1.0
Norfolk, Va.....	.0	.1	.1	.2	.2	.2	.5	.7	.8
Key West, Fla.....	.1	.3	.1	.4					
Pittsburgh, Pa.....	.1	.4	.0	.1	.4	.2	1.1	.9	.6
½-inch crushed coal—									
Portsmouth, N. H.....	.6			.4	.7	.9	1.3	1.6	1.4
Norfolk, Va.....	.1	.4	.3	.2	.5	.4	1.0	1.2	1.2
Key West, Fla.....	.0	1.0	.8	1.2					
Pittsburgh, Pa.....	.3	.5	.4	.6	.9	.6	1.6	1.6	1.7
Weathered:									
Run-of-mine coal—									
Portsmouth, N. H.....	.2	.4	.4	.2	.4	.7	1.1	1.0	1.4
Norfolk, Va.....	.0	.1	.0	.3	.2	.6	1.0	1.2	1.1
Key West, Fla.....	.5	.5	.6	.4	.8	1.2	2.6		
Pittsburgh, Pa.....	.1	.6	.0	.4	.5	.7	1.3	1.0	1.2
½-inch crushed coal—									
Portsmouth, N. H.....	.3	.3	.3	.5	.7	.9	1.3	1.2	1.3
Norfolk, Va.....	.1	.2	.3	.7	.7	1.2	1.5	1.8	1.3
Key West, Fla.....	.8	.4	.6	.7	1.8	2.1	2.8		
Pittsburgh, Pa.....	.3	.7	.6	.6	.8	1.1	1.6	1.6	1.3

TABLE 16.—Results of storage tests of New River coal showing ultimate composition of "unit" coal substance, and changes during storage.

	Carbon.	Hydrogen.	Nitrogen.	Oxygen.	Total sulphur.	Sulphate sulphur.
Mine sample 1, Aug. 27, 1909.....	89.86	4.87	1.51	2.76	0.88	0.006
Mine sample 2, Dec. 14, 1909.....	89.70	4.98	1.66	2.75	2.18	
Portsmouth, N. H.:						
As stored under water.....	89.63	4.89	1.53	4.07	1.06	.006
Submerged—						
2 years.....	89.71	4.66	1.74	2.87	1.04	.030
5 years.....	89.00	4.84	1.73	4.44	1.36	
As stored, exposed outdoors.....	89.63	4.86	1.57	2.75	1.24	.004
Exposed outdoors—						
3 months.....	89.86	4.70	1.54	2.90	1.35	.020
6 months.....	90.11	4.91	1.63	2.35	1.29	.038
1 year.....	89.91	4.69	1.58	2.89	1.18	
2 years.....	89.50	4.77	1.75	3.93	.98	.105
5 years.....	89.63	4.55	1.67	3.95	.76	
Exposed indoors 5 years.....	89.43	4.78	1.71	4.09	1.06	
Key West, Fla.:						
As stored under water.....	89.86	4.79	1.75	2.66	2.68	.011
Submerged—						
3 months.....	89.78	4.76	1.68	2.88	2.01	
2 years.....	90.30	4.58	1.59	2.19	2.96	
As stored, exposed outdoors.....	90.61	4.87	1.67	2.85	.79	.007
Exposed outdoors—						
3 months.....	89.66	4.70	1.62	4.00	1.29	
2 years.....	88.13	4.87	1.83	5.17	2.07	.26
Pittsburgh, Pa.:						
As stored under water.....	89.99	5.00	1.89	3.43	.71	.606
Submerged—						
3 months.....	90.18	4.79	1.64	3.39	.73	.000
6 months.....	90.26	4.85	1.63	3.26	.77	.005
1 year.....	90.15	4.68	1.68	3.49	.75	
As stored, exposed indoors.....	89.89	4.86	1.55	4.00	1.25	.005
Exposed indoors—						
3 months.....	89.93	4.79	1.66	3.63	1.23	.020
6 months.....	89.36	5.17	1.65	3.52	1.35	.022
1 year.....	90.06	4.67	1.63	3.62	1.26	
2 years.....	90.31	4.74	1.70	3.25	1.10	.043
Exposed outdoors, 5 years.....	90.23	4.75	1.74	3.25	.76	.06

DISCUSSION OF RESULTS OF TESTS.

The tables show that the maximum deterioration in heating value in one year was 1.2 per cent in one-fourth inch coal exposed indoors at Key West (Dry Tortugas); and the maximum in two years was 2.1 per cent in the same portion after a second year out of doors. The warmer climates of Norfolk and Key West (Dry Tortugas) caused uniformly greater deterioration in the exposed coal than the cooler ones of Pittsburgh and Portsmouth. The one-fourth inch coal showed losses almost always 50 to 100 per cent greater than those of the run-of-mine.

After one year the submerged coal in six out of eight portions at the different storage points showed no loss of heat value. Two portions had lost about 0.4 per cent. After two years' submergence, three out of eight portions showed losses ranging from 0.4 to 0.6 per cent, the remainder being less than 0.2 per cent, and after five years the loss in two portions was practically nothing, the other two tested showing losses of 0.4 and 0.5 per cent.

Slight, continuous decreases in heating value were noted after the two-year period in the exposed coal, the maximum being 2.6 per cent at Key West (Dry Tortugas). In five years the maximum loss at Portsmouth was 1.4 per cent, at Pittsburgh 1.7 per cent, and at Norfolk 1.3 per cent.

It should be noted that the probable experimental accuracy in the determination of calorific value by the method used was not more than 0.2 per cent, and that, including the effect of the errors in determination of moisture, ash, and sulphur, in computing the "unit coal" value, a possible deviation of 0.4 per cent might occur between the values obtained for duplicate samples of the same portion. The losses indicated, therefore, in the submergence tests are almost within the experimental error, and may be said to have been inconsiderable. The tests of coal in box 10 at Norfolk appear to show that alternate drying in the open air for brief intervals between the submergence periods causes some loss, although not a material one.

The variation in ultimate composition of the coal substance as shown in Table 16 was very slight, if any. The probable experimental error in these analyses is 0.25 per cent for carbon, 0.03 per cent for hydrogen, 0.02 per cent for nitrogen, and 0.20 per cent for moisture and ash, throwing an aggregate probable error on the oxygen (determined by difference) of 0.50 per cent, or a possible variation between duplicates of 1 per cent. Any indications of change in percentage of oxygen in the coal during storage are, therefore, within the experimental error, and the only conclusion justifiable is that the change in oxygen content, if any, appears to be of the same order of magnitude as the change in calorific value.

No actual determination of the amount of physical deterioration of lumps was made, but by observation of the run-of-mine coal stored outdoors at Pittsburgh it was noted that very little slacking or weakening of the lumps occurred during two years. This is illustrated in Plate IV, *A*, which shows the condition of the run-of-mine coal after two years' exposure. Plate IV, *B*, shows crushed coal after two years' exposure under similar conditions.

CONCLUSION.

In general, the conclusion to be drawn from these tests is that New River coal, under severe conditions of outdoor exposure to the weather, deteriorates in heating value approximately 1 per cent in the first year, 2 per cent in the first two years, and not over 3 per cent in five years. Storage under water prevents practically all deterioration during one year, and no more than 0.5 per cent has been found in any test for two years or less. Salt water possesses no advantage over fresh water in preventing deterioration. Intermittent exposure and partial drying of the submerged coal probably causes deterioration in some degree, although very small.

Submergence storage of New River coal is not to be recommended for the sake of preventing deterioration in heat value. Its advantage lies only in insuring against spontaneous combustion.

TESTS OF PITTSBURGH GAS COAL AT ANN ARBOR, MICH.

Experiments to determine the loss in heating value of coal of the Pittsburgh bed and its deterioration for purposes of illuminating-gas manufacture, during periods of storage ranging from six months to five years, were begun in November, 1910, at Ann Arbor, Mich., in cooperation with the Michigan Gas Association and the University of Michigan. The periodic sampling of the coal for determination of its heating value was done under the direction of Prof. A. H. White, of the university, who also carried out the tests of the coal for yields of gas and by-products. The results of the gas tests are to be reported by Prof. White.

Coal was obtained for the tests from two different mines, both working the Pittsburgh bed and producing coal commonly used in gas manufacture.

SOURCE OF THE COAL.

The coal designated in the tests "AA15" was mined November 1, 1910, at the Schoenberger mine at Baird Station, Washington County, Pa. It consisted of a small carload (21½ tons, net weight) of lump coal screened over a three-fourths-inch bar screen, and was loaded and sampled under the supervision of one of the mining engineers of the bureau. Mine samples were taken at six different points in the mine and a



**A. RUN-OF-MINE NEW RIVER COAL AFTER TWO YEARS' WEATHERING AT PITTSBURGH, PA.
(SHOWING LUMPS INTACT).**



B. FINELY CRUSHED NEW RIVER COAL AFTER TWO YEARS' WEATHERING AT PITTSBURGH, PA.



composite of these made for analysis. The car arrived at Ann Arbor November 12, 1910, and was unloaded into the bins November 15 and 16, a gross sample of 3 to 4 tons representing the car being set aside at that time. On November 18 and 19 this gross sample was reduced to two 300-pound lots as described in page 24, and on November 23 these were further reduced for analysis to two laboratory samples of 2½ to 3 pounds each.

The coal designated "AA16" was mined December 9, 1910, at Consolidation No. 63 mine at Monongah, Marion County, W. Va. It was a small carload (29½ tons, net weight) of coal screened over a three-fourths-inch barscreen, loaded and sampled under the supervision of a mining engineer of the bureau. Mine samples were taken at four different points in the mine and a composite of these made for analysis. The car arrived at Ann Arbor December 28, 1910, and was unloaded into the bins December 30 and 31. A gross sample of 3 or 4 tons was removed during the unloading and reduced at that time, as described elsewhere (p. 24), to two 300-pound lots. On January 6, 1911, these were crushed and reduced further to laboratory samples.

TESTS IN OPEN BINS.

The bins were located at the works of the Ann Arbor Gas Co. in the open yard adjoining the retort house. The coal in the bins was exposed to the weather, the bins being so constructed that the conditions in an ordinary coal pile 10 feet deep were approximately simulated. Each of the two bins covered a floor space 11½ by 7½ feet, was 10 feet high, and was divided by board partitions into six compartments. Each compartment was 7½ feet long, 10 feet high, and 20 inches wide, held about 3 tons of coal at 7 feet depth and 4 tons at 9½ feet depth. The bins were backed up against a concrete wall, had a board floor, and were uncovered except for a screen of poultry netting.

At the end of each successive period one of the six compartments was emptied and the coal tested, both for heating value and for gas-making qualities.

TESTS OF STORAGE BY SUBMERGENCE UNDER WATER.

Four barrels, each holding 300 to 320 pounds, were filled, two with each kind of coal. Both ends of each barrel were perforated with about 10 ¼-inch holes to allow escape of air and complete filling with water when submerged.

The barrels of coal "AA15," numbered 1 and 2, were filled with coal (part of the 3 or 4 ton gross sample above referred to) on January 7, 1911, and were submerged January 14, 1911; those of coal "AA16," numbered 3 and 4, were filled January 9 and submerged January 14.

The barrels were lowered under the water of a supply basin (see Pl. V, A, p. 32) 6 or 8 feet deep, near the gas works, the water of which is kept fresh constantly by springs and does not freeze.

One of the barrels of each kind of coal remained submerged continuously until the expiration of $5\frac{1}{2}$ years of storage. The other was raised after the expiration of 6 months, $1\frac{1}{2}$ years, $2\frac{1}{2}$ years, $3\frac{1}{2}$ years, and $4\frac{1}{2}$ years, sampled each time, and returned to the water.

METHODS OF PREPARING AND SAMPLING COAL.

The cars of coal were unloaded by hand, single shovelfuls being thrown in rotation into each of the six bins and into a chute leading to an inclosed sampling floor. The carload was thus divided into seven equal representative portions. The portion on the sampling floor, 3 or 4 tons, was, without further preparation, divided into four equal portions, three of these being sacked for gas tests, and the fourth reserved for sampling. By alternate shovelfuls this last quarter was divided into two duplicate portions of 800 to 1,000 pounds each, and these in turn were each divided into five equal portions. Of these five portions, of 160 to 200 pounds each, one served for a special "change of weight" test in an open ash-can, two of the others made up the barreled lot for submergence, and the last two portions, aggregating 300 to 400 pounds, served for reducing down to laboratory samples. This was done by crushing the coal and quartering it in the usual manner by spreading out and rejecting opposite quarters. After each reduction of quantity the coal was crushed to a smaller size.

TESTS TO DETERMINE CHANGE IN WEIGHT DURING STORAGE.

In order to determine the change in weight of the dry coal substance by oxidation or other weathering effect, during storage, two weighed portions of each coal of about 200 pounds each were placed in galvanized-iron ash cans, without covers, and exposed to outdoor conditions. The bottoms of these cans were perforated to allow water to drain out, the coal resting on two thicknesses of 16-mesh wire screening so as to reduce mechanical losses to a minimum. Similar screening covered the tops of the cans. After standing indoors for two or three weeks, to become air-dried, the cans were weighed and then buried to half their depth in the coal of the test bins. After stated periods of exposure the cans were air dried, weighed, sampled, again weighed, and returned to the bins. By computing from the analysis and weight the dry coal substance present, any change of weight due to weathering was determined. The weighings were made with an accuracy of 0.5 pound or about 0.2 per cent.

The results of these tests indicate little, if any, change of weight within the error of measurement of the dry coal substance in five years' exposure. Table 17 shows the actual weights.

TABLE 17.—*Change in weight of Pittsburgh gas coal during five years' exposure.*

Item.	Coal AA15, from Baird Station, Pa. (can 2).		Coal AA16, from Monongah, W. Va. (can 4).	
	As stored.	After five years.	As stored.	After five years.
Gross weight, pounds.....	239.0	238.5	237.5	235.0
Weight of can, pounds.....	30.5	30.5	30.0	30.0
Net weight of coal, pounds.....	208.5	208.0	207.5	205.5
Moisture in coal, per cent.....	2.2	1.6	2.9	1.7
Weight of dry coal, pounds.....	203.9	204.7	201.5	202.0
Change in dry weight, per cent.....		+0.4		+0.3

The recorded weights of cans 1 and 3, taken at intervals during the five years' period, are incomplete and have been omitted from the table, because, unfortunately, a record was not preserved in every instance of the weights both before and after sampling so as to show how much coal was lost in sampling. The final change in weight of these portions therefore represents both the loss in sampling and the change due to weathering, and the latter is indeterminate.

The results showing deterioration in heating value of the two coals, both in the open bins and in the submergence tests, are given in Tables 18 and 19. It is to be noted, as was explained in connection with the tests of New River coal, that the only fair basis for comparing analyses in determining deterioration is the dry coal substance, free of sulphur and ash.

On this basis the amount of deterioration in one year's open air storage was practically negligible, even in the upper six inches of the exposed coal. During the second, third, fourth, and fifth years the deterioration proceeded very slowly and did not reach an amount greater than 1.1 per cent in five years. The submerged portions may be said to have suffered practically no loss measureable by the degree of accuracy used.

TABLE 18.—Storage tests of screened lump (over 4-inch) coal from the Pittsburgh bed at Baird Station, Pa., exposed to weather at Ann Arbor, Mich., in 3-ton lots.

Weight under test.	Method of storing.	Sample from—	Date of sampling.	Duration of storage.	Number of samples averaged.	Moisture.	Analysis on dry basis.					Heating value of "unit coal."		
							Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Calo-ries.	B. t. u.	Calo-ries.	Loss in B. t. u.
		Mine.	Nov. 3, 1910			Per ct.	Per ct.	Per ct.	Per ct.	Per ct.	7,991	14,384	8,545	Per ct.
		Car during loading.	Nov. 1, 1910		4	2.44	34.93	59.14	6.30	1.06	7,968	14,343	8,549	
		Car during unloading.	Nov. 23, 1910		2	2.23			6.34	.89	7,959	14,325	8,541	
3 tons.	Open bin.....	Surface of bin.	June 8, 1911	6 months.	2	2.41			8.29	.90	7,782	14,008	8,539	15,370
	do.	Surface of bin.	do.	1 do.	2	2.00			7.13	.68	7,885	14,211	8,531	15,392
	do.	Surface of bin.	Dec. 4, 1911	1 year.	2	6.34			8.47	.96	7,752	13,944	8,526	15,347
	do.	Surface of bin.	Dec. 19, 1912	2 years.	2	8.59			8.16	1.05	7,782	13,068	8,508	15,350
	do.	Surface of bin.	do.	do.	2	2.38	34.37	58.14	7.40	1.07	7,817	13,071	8,471	15,263
	do.	Surface of bin.	Nov. 3, 1913	3 years.	2	2.13	34.09	57.92	7.18	1.02	7,727	13,908	8,477	15,268
	do.	Surface of bin.	Oct. 31, 1913	do.	2	8.80	36.85	54.51	8.24	1.03	7,716	13,880	8,497	15,287
	do.	Surface of bin.	Dec. 11, 1914	4 years.	1	2.72	36.84	57.60	6.26	1.08	7,857	14,143	8,493	15,286
	do.	Surface of bin.	Jan. 29, 1916	5 years.	1	2.45	35.40	58.09	6.55	1.12	7,876	14,177	8,479	15,262
	do.	Surface of bin.	do.	do.	1	1.88	35.35	58.09	6.37	1.01	7,902	14,224	8,477	15,259
	do.	Surface of bin.	do.	do.	1	1.55	35.46	58.27	6.52	.89	7,998	14,366	8,533	15,369
300 pounds.	Submerged.	Barrel 1	Jan. 7, 1911	As stored.	2	2.09			5.84	.97	8,015	14,427	8,501	
	do.	do.	June 15, 1911	6 months.	1	1.21			6.60	1.13	7,885	14,192	8,501	15,302
	do.	do.	May 29, 1912	14 years.	2	4.42	35.23	58.09	7.25	1.20	7,828	14,080	8,499	15,298
	do.	do.	June 9, 1913	24 years.	1	6.04	35.74	57.01	6.30	1.12	7,947	14,305	8,540	15,372
	do.	do.	June 8, 1914	24 years.	1	3.72	36.64	56.97	6.36	1.06	7,740	13,932	8,559	15,406
	do.	do.	May 19, 1915	44 years.	1	2.47	35.12	55.95	8.93	1.06	7,740	13,932	8,559	15,406
	do.	do.	do.	do.	1	2.47	36.31	57.17	6.52	1.31	7,919	14,264	8,527	15,349
	do.	do.	do.	do.	1	2.47	36.31	57.17	6.52	1.31	7,919	14,264	8,527	15,349

a Galn.

TABLE 10.—Storage tests of screened lump (over 3-inch) coal from the Pittsburgh bed at Monongah, W. Va., exposed to the weather at Ann Arbor, Mich.; in 4-ton lots.

Weight under test.	Method of storing.	Sample from—	Date of sampling.	Duration of storage.	Number of samples averaged.	Mds- ture.	Analysis on dry basis.						Heating value of "unit coal."		
							Volatiles matter.	Fixed carbon.	Ash.	Sulphur.	Calo-ries.	B. t. u.	Calo-ries.	B. t. u.	Loss in B. t. u.
						Per ct.	Per ct.	Per ct.	Per ct.	Per ct.					Per ct.
4 tons.	Open bin.	Mine.	Dec. 10, 1910		4	2.96	36.07	58.16	6.77	0.69	7,935	14,283	8,458	15,224	0.2
Do.	do.	Car during loading.	do.		2	2.35			7.78	.68	7,906	14,060	8,500	15,316	
Do.	do.	Car during unloading.	Jan. 6, 1911			2.91			9.04	.68	7,681	13,827	8,514	15,323	
Do.	Open bin.	Entire bin.	June 8, 1911	6 months.	2	2.22			8.85	.74	7,726	13,907	8,528	15,350	
Do.	do.	Surface of bin.	do.	do.	1	2.16			8.90	.66	7,711	13,880	8,513	15,323	
Do.	do.	Entire bin.	Dec. 16, 1911	1 year.	2	3.19			9.24	.78	7,680	13,833	8,516	15,329	0
Do.	do.	Surface of bin.	do.	do.	1	4.56			9.19	.82	7,674	13,814	8,507	15,313	1
Do.	do.	Entire bin.	Dec. 19, 1912	2 years.	2	2.40			9.68	.88	7,593	13,667	8,468	15,237	6
Do.	do.	Surface of bin.	do.	do.	1	2.41	85.77	54.56	7.70	.77	7,742	13,906	8,443	15,197	8
Do.	do.	Entire bin.	Nov. 3, 1913	3 years.	2	3.45	36.35	55.72	7.70	.72	7,770	13,966	8,464	15,235	0
Do.	do.	Surface of bin.	do.	do.	1	2.46	35.04	56.80	8.16	1.12	7,763	13,973	8,511	15,220	0
Do.	do.	Entire bin.	Dec. 11, 1914	4 years.	2	2.41	36.55	55.61	7.54	.81	7,841	14,114	8,467	15,241	5
Do.	do.	Surface of bin.	do.	do.	1	3.26	37.25	55.80	6.92	.70	7,812	14,062	8,455	15,219	7
Do.	do.	Entire bin.	Jan. 29, 1916	5 years.	1	1.93	37.06	55.88	7.11	.65	7,732	13,918	8,419	15,149	1.1
Do.	do.	Surface of bin.	do.	do.	1	1.73	37.47	54.58	9.10	.72	7,666	13,799	8,506	15,311	
Do.	do.	Entire bin.	Jan. 9, 1911	As stored.	2	2.81			8.23	.69	7,776	13,966	8,521	15,338	2
Do.	Submerged.	Barrel 3.	June 15, 1911	6 months.	1	1.41			8.23	.75	7,764	13,967	8,453	15,215	6
Do.	do.	do.	May 29, 1912	14 years.	1	3.00	36.54	55.74	7.72	.76	7,810	14,046	8,479	15,262	3
Do.	do.	do.	June 9, 1913	24 years.	1	4.75	37.98	54.75	7.86	.87	7,630	13,734	8,524	15,243	2
Do.	do.	do.	June 8, 1914	34 years.	1	3.25	37.31	52.83	9.84	.83	7,683	13,829	8,521	15,338	2
Do.	do.	do.	May 19, 1915	44 years.	1	2.16	38.01	52.75	9.24	.88	7,683	13,829	8,521	15,338	2
Do.	do.	do.	do.	do.	1	2.60	37.71	53.63	8.66	.81	7,692	13,846	8,474	15,253	4
Do.	do.	Barrel 4.	do.	do.	1										

s Gain.

TABLE 20.—*Ultimate composition of coals from the Pittsburgh bed, on basis of ash and moisture free substance, after various periods of storage at Ann Arbor, Mich.*

	"AA15" coal from Baird, Pa.					"AA16" coal from Monongah, W. Va.				
	Carbon.	Hydrogen.	Nitrogen.	Oxygen.	Sulphur.	Carbon.	Hydrogen.	Nitrogen.	Oxygen.	Sulphur.
Mine sample (composite).....	Per ct. 85.44	Per ct. 5.41	Per ct. 1.52	Per ct. 6.48	Per ct. 1.15	Per ct. 85.17	Per ct. 5.47	Per ct. 1.51	Per ct. 7.12	Per ct. 0.73
Car sample, as unloaded.....	85.09	5.32	1.65	6.79	1.15	84.84	5.67	1.68	6.88	.93
Coal in open bins after 4 years' weathering.....	84.91	5.47	1.61	6.94	1.10	85.01	5.63	1.74	6.75	.86
Coal in open bins after 5 years' weathering.....	85.27	5.56	1.70	6.27	1.20	84.86	5.61	1.68	7.04	.81
Same, 6-inch surface layer.....	84.94	5.46	1.71	6.81	1.08	84.95	5.74	1.75	6.88	.68
Coal, as submerged for test.....	85.35	5.54	1.68	6.41	1.02	84.36	5.49	1.63	7.57	.96
Same after 4½ years' storage, submerged.....	85.61	5.64	1.66	5.69	1.40	84.84	5.68	1.73	6.86	.89

TESTS OF POCAHONTAS COAL MADE ON THE ISTHMUS OF PANAMA.

During the year ended June 1, 1910, a large number of samples were taken of Pocahontas (Va.) coal as unloaded on one part of the general stock pile of the Panama Railroad Co. at dock No. 14, Cristobal, Isthmus of Panama. These samples were analyzed in order to determine the average heating value of the coal, as placed on the pile, for comparison with the heating value of the same coal as determined on removal from the pile after different periods of storage. Some time previous to June 1, 1910, when the fact was discovered by the Bureau of Mines, this part of the stock pile was several times almost entirely dug up and consumed, and no samples of the coal were taken. The analyses made up to that time became, therefore, useless for any test of deterioration, and a new series of tests was begun.

On June 16, 1910, a small test pile of 120 tons of Pocahontas run-of-mine was established near dock No. 14, separate from the general stock pile. The coal was the cargo of the steamship *Vauxhall*, leaving Norfolk, Va., June 1, and unloaded at Cristobal June 13 to 18. One average sample of the coal as placed on the pile was sent to the bureau for analysis, but no report was made to the bureau of the exact method of taking the sample. Every three months after the test pile was established a 10-ton portion (consisting of an entire vertical section across the shorter dimension of the pile) was removed and thoroughly sampled. Eight samples were taken each time, by throwing aside 400 to 800 pounds into each of four boxes, in small portions at regular intervals as the 10 tons were removed; these 400-pound portions were then crushed and reduced by quartering, two small can samples being taken from each.

A summary of the results follows:

TABLE 21.—*Storage test of Pocahontas (Va.) coal.*

[120 tons in open pile, near dock No. 14, Cristobal, Isthmus of Panama.]

Item.	Original as stored, June 16, 1910 (sample).	After 3 months' storage, Sept. 21, 1910 (average of 7 samples).	After 6 months' storage, Dec. 18, 1910 (average of 8 samples).	After 9 months' storage, Mar. 20, 1911 (average of 8 samples).	After 13 months' storage, July 18, 1911 (average of 8 samples).	After 2 years' storage, June 16, 1912 (average of 2 samples).
Air-drying loss.....		4.37	6.28	3.89	4.21	5.55
Coal, as received:						
Moisture.....	0.94	4.98	6.78	4.47	5.34	6.79
Ash.....	5.51	6.78	6.66	5.53	6.20	6.56
Sulphur.....	.78	.49	.52	.50	.48	.57
Calories.....	8,188	7,731	7,625	7,764	7,716	7,508
British thermal units.....	14,738	12,866	12,726	12,975	13,889	13,515
Coal, free of moisture, sulphur and ash (calculated):						
Calories.....	8,794	8,786	8,761	8,768	8,762	8,723
British thermal units.....	15,829	15,815	15,770	15,779	15,772	15,701
Percentage of loss (in B. t. u.) during storage.....		0.09	0.28	0.32	0.37	0.81

* For a check on this analysis of coal as stored, the analysis of 14 samples taken from the same cargo when loaded at Norfolk, June 1, 1910, show on the moisture, sulphur, and ash-free basis an average calorific value of 8,790 calories.

As explained under the report on New River coal, the only fair basis of comparison in studying deterioration of heating value is the coal substance free of its accidental impurities—moisture, sulphur, and ash.

On this basis, therefore, and on the basis of average samples of the entire cross-section of the pile, the results given in Table 21 show that during one year's outdoor exposure this coal deteriorated very slightly (less than 0.4 per cent) in heating value, and that the deterioration took place entirely during the first six months (June 15 to Dec. 15). There was a further deterioration of 0.4 per cent during the second year.

The climatic conditions during the period of storage are indicated by the following table:

TABLE 22.—*Monthly temperature averages and precipitation at Colon, R. P., June 1, 1910, to June 1, 1912.*

	Temperature of air.			Average tempera- ture of sea water.	Total precipita- tion.
	Average mean.	Average maximum.	Highest.		
1910.	° F.	° F.	° F.	° F.	Inches.
June.....	79.4	83.9	89	83.1	12.63
July.....	77.6	82.1	86	82.5	21.07
August.....	78.4	82.5	86	83.2	14.98
September.....	78.6	84.0	88	82.6	12.05
October.....	78.5	84.4	89	82.5	15.65
November.....	77.8	82.1	88	80.6	30.04
December.....	77.3	80.5	82	79.8	15.20
1911.					
January.....	78.4	81.4	82	80.2	0.99
February.....	77.9	81.1	82	80.3	1.81
March.....	78.3	81.5	82	80.3	1.41

TABLE 22.—*Monthly temperature averages and precipitation at Colon, R. P., June 1, 1910, to June 1, 1912—Continued.*

	Temperature of air.			Average temperature of sea water.	Total precipitation.
	Average mean.	Average maximum.	Highest.		
1911.					
April.....	° F. 79.6	° F. 82.8	° F. 84	° F. 81.6	<i>Inches.</i> 3.06
May.....	79.5	84.8	88	82.6	17.13
June.....	79.3	83.7	88	82.4	16.58
July.....	81.1	84.4	86	84.0	14.58
August.....	79.7	83.6	86	84.1	11.60
September.....	80.6	84.7	90	84.8	11.62
October.....	79.4	83.6	90	81.6	16.53
November.....	79.4	84.3	89	81.9	15.81
December.....	82.0	86.4	89	81.7	2.63
1912.					
January.....	82.1	86.4	88	80.9	0.28
February.....	80.8	84.7	88	80.7	1.81
March.....	82.5	86.8	88	81.1	0.66
April.....	82.8	87.7	90	82.8	0.75
May.....	82.0	86.6	91	83.4	12.03

TESTS OF SHERIDAN (WYO.) SUBBITUMINOUS COAL.

About December 30, 1907, five wooden bins, constructed by the Chicago, Burlington & Quincy Railway Co. in the railroad yards at Sheridan, Wyo., were filled with coal from the Dietz mines near Sheridan. The bins adjoined one another in the same structure, the side walls of each serving as partitions. They were built of heavy matched lumber, but allowed more or less circulation of air through cracks. One bin was left without roof, and the coal thus fully exposed to the weather, but the other four were roofed over.

Bin 1, 4 feet wide, 8 feet high, with open top and end, contained about 4 tons of run-of-mine coal piled about 5 feet deep.

Bin 2 was a duplicate of bin 1, except that top and ends were closed.

Bin 3, of the same size as bins 1 and 2; with closed top and ends, contained about 4 tons of run-of-mine coal, moistened with about 7 per cent of added water.

Bin 4, 4 feet wide, 18 feet high, with closed top and ends, contained 10 to 12 tons of run-of-mine coal, piled about 15 feet deep.

Bin 5 was a duplicate of bin 4, except that slack coal (through 3-inch mesh screens) was used instead of run of mine.

Bins 1, 2, 3, and 4 were loaded with run-of-mine coal from one carload, and three samples were taken from the car as it was unloaded. Small portions were thrown aside at regular intervals during the unloading, and the three gross samples thus obtained were crushed to one-half-inch size and quartered down. Bin 5 was filled from a car of the commercial output of slack coal (3-inch screenings), and one sample was taken by the method just described.

All sampling after that at the start of the tests and prior to that at the expiration of two and three-fourths years was done by the so-

called "grab" method, so as not to disturb the entire lot, and thus expose its under portions to undue weathering. A spot 2 by 3 feet square was dug away for about 1 foot in depth and several small, well-distributed portions removed therefrom. These were combined, crushed, mixed, and quartered to the laboratory size.

The sampling carried out October 1, 1910, after a two and three-fourths years' period, was done by rehandling the entire amount in each bin. The coal was transferred to a temporary bin, and during the transfer an average sample was taken by the method used when loading the bins. The coal was then returned to its proper bin.

The samples were mailed to the Pittsburgh laboratory in sealed metal cans and there analyzed. The "unit coal" basis, as previously explained (p. 12), was used in all comparisons of calorific values.

It was impracticable in these tests to determine accurately the change in weight of the actual fuel substance of the coal, and in fact an element of uncertainty due to the same cause enters into practically all tests of deterioration of the coal. Laboratory experiments have shown that coal ordinarily increases slightly in weight on exposure to the air, if the measurement be made on the basis of actual fuel substance. It is possible, therefore, that the net losses in heating value may be slightly less than are reported, since the actual weight of fuel substance present may be somewhat greater, although its heat value is less than when the coal was stored.

TABLE 23.—Loss in heat value of Sheridan (Wyo.) coal during storage.

	As received.					"Unit coal" basis.		Loss in B. t. u.
	Moisture.	Ash.	Sulphur.	Calorics.	B. t. u.	Calorics.	B. t. u.	
Bin 1—open:	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>					<i>Per cent.</i>
As stored.....	21.38	7.77	1.11	5,181	9,326	7,370	13,266	
After 3 months.....	21.06	9.83	1.06	4,915	8,847	7,174	12,913	2.86
After 9 months.....	20.39	8.63	.88	4,963	8,968	7,070	12,726	4.07
After 2½ years.....	18.75	7.24	.76	5,250	9,450	7,135	12,843	3.19
Bin 2—closed:								
As stored.....	21.38	7.77	1.11	5,181	9,326	7,370	13,266	
After 3 months.....	20.92	10.36	.81	4,907	8,833	7,202	12,964	2.28
After 9 months.....	17.57	10.00	1.08	5,034	9,061	7,010	12,618	4.88
After 2½ years.....	15.32	7.96	.86	5,420	9,756	7,064	12,769	3.76
Bin 3—closed, coal moistened:								
As stored.....	21.38	7.77	1.11	5,181	9,326	7,370	13,236	
After 3 months.....	22.00	9.59	1.00	4,821	8,678	7,109	12,796	3.54
After 9 months.....	18.86	7.90	1.01	5,129	9,232	7,053	12,695	4.30
After 2½ years.....	15.62	9.70	.90	5,244	9,439	7,076	12,737	3.99
Bin 4—deep, closed:								
As stored.....	21.38	7.77	1.11	5,181	9,326	7,370	13,236	
After 3 months.....	20.38	9.00	.91	5,116	9,209	7,303	13,145	.91
After 9 months.....	18.78	6.56	.80	5,221	9,384	7,033	12,659	4.57
After 2½ years.....	12.53	9.84	.94	5,361	9,686	6,982	12,568	5.26
Bin 5—closed, deep, screenings:								
As stored.....	20.82	11.62	1.21	4,915	8,947	7,355	13,199	
After 3 months.....	20.65	10.63	.88	4,582	8,788	7,166	12,899	2.57
After 9 months.....	18.21	10.87	.94	4,974	8,953	7,066	12,719	3.93
After 2½ years.....	16.20	11.99	.92	4,975	8,955	6,990	12,582	4.96

Three mine samples were taken from the Dietz mines at different times, one from No. 2 mine, April 9, 1909, one from No. 4 mine,

November 22, 1911, and one from No. 2 mine, November 22, 1911. The analyses of these three mine samples are given below for comparison with those of the car samples taken when the coal was stored. These samples represent the average commercial output of the mines. The interval of two and one-third years between these samples gives a check on the uniformity of the sampling and analytical methods.

TABLE 24.—*Analyses of samples of coal from Dietz mines, Sheridan, Wyo.*

Mine from which sample was taken.	As received.					"Unit coal" basis. ^a	
	Moisture.	Ash.	Sulphur.	Calories.	B. t. u.	Calories.	B. t. u.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>				
No. 2.....	23.56	4.98	0.74	5,232	9,418	7,387	12,261
No. 2.....	21.64	8.99	1.28	5,056	9,104	7,358	12,244
No. 4.....	22.85	8.46	1.18	4,986	8,973	7,219	12,174

^a Moisture, sulphur, and ash-free basis.

After nine months (Sept. 1, 1908), a photograph was taken of bins 1 and 2, showing the effect of the closed bin as compared with the open in preventing slacking of lumps. (See Pl. V, B.) After two and three-fourths years (September, 1910) the coal in each bin was photographed at rather close range, before and after removal from the bins, so as to show its physical appearance both on the surface and after being thoroughly mixed by rehandling. (See Pls. V, C, VI, and VII.) Careful observations also were made at this time to determine the extent of the slacking in each bin.

Bin 1 (open, coal piled 5 feet deep): A fairly uniform layer, 8 inches to 12 inches thick, of closely packed, finely disintegrated coal (see Pl. VI, A) had formed over the surface. Below that the coal was practically in the same condition as when stored. Near the center, especially, the lumps were harder and brighter than at the sides, and harder also than those in bins 2, 3, and 4. (See Pl. VII.)

Bin 2 (closed, coal piled 6 feet deep): Coal had slacked somewhat on the surface (Pl. VI, B), but had not completely disintegrated, as had coal in bin 1. The lumps all through the bin were of a dull color and showed more or less cracking. (See Pl. VII, B.) They broke up badly on handling.

Bin 3 (closed, 6 feet deep, coal moistened): Lumps had cracked and weakened more than in bin 2; otherwise appearance was nearly the same.

Bin 4 (closed, run-of-mine coal, 15 feet deep): Physical appearance of coal in the upper two-thirds of bin was much the same as in bin 2, that is, the lumps were still more or less intact, but were cracked and weakened so that they broke up badly on handling. The coal in the lower third of the pile seemed to be somewhat brighter and firmer.



A. METHOD OF SUBMERGING BARRELS OF PITTSBURGH COAL AT ANN ARBOR, MICH.



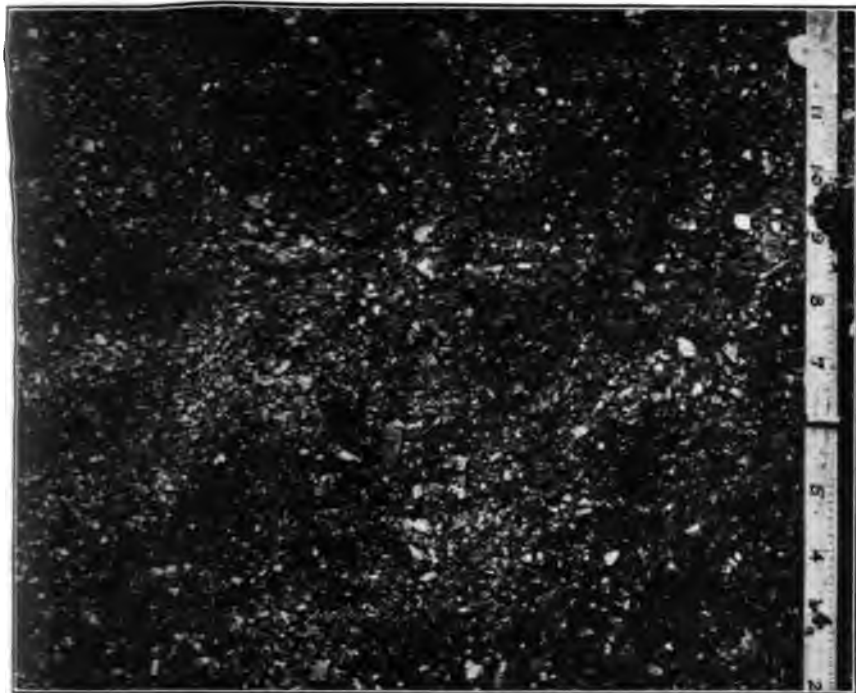
B. SUBBITUMINOUS COAL STORED AT SHERIDAN, WYO., AFTER 2¾ YEARS.

Bin 1 on right, bin 2 on left. Note effect of closed bin on preserving lumps.



C. SUBBITUMINOUS COAL FROM BIN 1, AFTER 2¾ YEARS' STORAGE AT SHERIDAN, WYO.

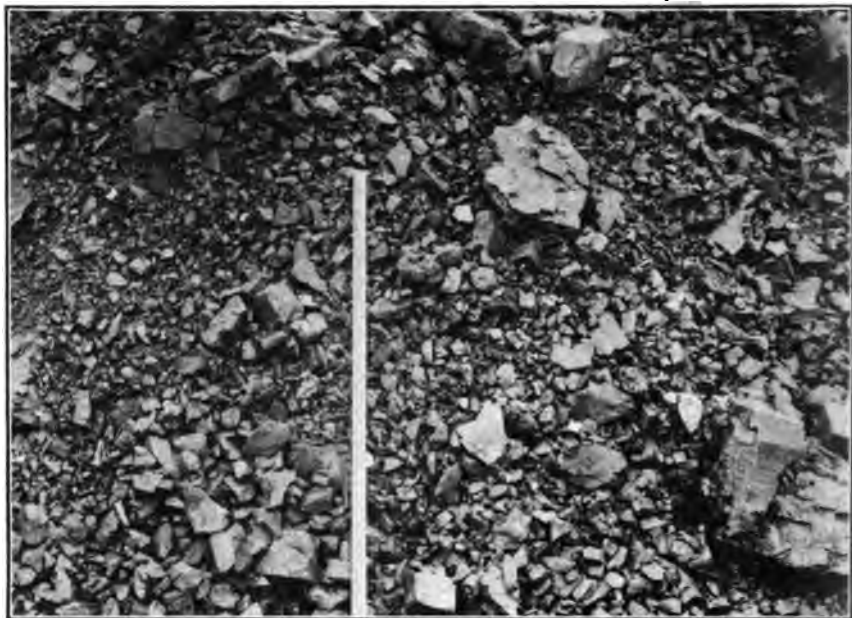




A. SUBBITUMINOUS COAL IN BIN 1, AT SHERIDAN, WYO., AFTER $2\frac{3}{4}$ YEARS' STORAGE.



B. SUBBITUMINOUS COAL IN BIN 2, AT SHERIDAN, WYO., AFTER $2\frac{3}{4}$ YEARS' STORAGE.



A. NEAR VIEW OF COAL FROM BIN 1, AFTER REMOVAL. NOTE SIZE OF LUMPS. (SHERIDAN, WYO.)



B. NEAR VIEW OF COAL FROM BIN 2, AFTER REMOVAL. NOTE SIZE OF LUMPS. (SHERIDAN, WYO.)

Bin 5 (closed, screenings, 15 feet deep): Coal apparently was less weathered than in bin 4; seemed to have been very little altered in lower two-thirds of pile.

Referring to the results in Table 23, during the first three months, the greatest loss, 3.54 per cent, was in the shallow bin, bin 3, where the coal had been moistened. Then followed, in order, bin 1, 2.66 per cent; bin 5, 2.57 per cent; bin 2, 2.28 per cent; and bin 4, 0.91 per cent. The added moisture seemed to increase the deterioration. After nine months, however, the order was changed, bin 2, the closed, shallow bin, showing the greatest loss, 4.88 per cent; and following that, in order, bin 4, 4.57 per cent; bin 3, 4.30 per cent; bin 1, 4.07 per cent; bin 5, 3.93 per cent. During the hot, dry weather of summer, the deterioration had been more or less equalized in all the bins. Bin 1, the open bin, seems to have been protected by its surface layer of slack fully as well as the others were by the roof of the bins.

The seemingly greater loss of heat value at this nine months' stage in bins 1, 2, and 3 than was shown two years later in the same bins is easily explained by the fact that the nine months' samples were grab samples taken 8 to 12 inches below the surface, where greater weathering could occur, whereas the 2½-year samples were representative of the entire lot in each bin and included parts from the interior that were scarcely weathered at all.

After 2½ years bin 4, the deep closed bin, showed the greatest loss, 5.26 per cent; then followed, in order, bin 5, 4.96 per cent; bin 3, 3.99 per cent; bin 2, 3.75 per cent; and bin 1, 3.19 per cent.

CONCLUSIONS.

Evidently Sheridan coal under the conditions of these tests loses 3 to 5.5 per cent of its heat value in about three years' storage, the greater part (70 to 80 per cent) of this loss being in the first nine months. During the period of 2½ years the deep bins suffered the greatest loss, probably because their sides offered greater surface for access of air than those of the small bins. The latter became covered with a 12-inch layer of fine slack that helped to protect the layers beneath from oxidation. In the deep bins, the lumps became badly cracked, but retained their form sufficiently to give more ready access of air, and thus permit greater oxidation.

In the storage of Sheridan coal for more than three months, covering the bins is not as advantageous as the use of air-tight bottoms and sides (of concrete, for example), and the accumulation of a protecting layer of fine slack on the surface. The deterioration of Sheridan coal in heat value can probably in this manner be kept below 3 per cent in one year, and will probably not increase to more than 4 per cent in two or three years if the coal remains undisturbed.

Physical deterioration (slacking) is also largely prevented in the under portions by the formation of a closely packed layer of slack, at least 12 inches thick on the surface.

Although no indications of spontaneous heating were noted in the tests herein described, it is found in practice to be dangerous, on account of dangerous heating, to store Sheridan coal in piles greater than about 10 feet in depth or width. In large masses of coal radiation of spontaneously developed heat is restricted to a dangerous degree. Submergence under water would probably prove particularly advantageous as a means of safely storing subbituminous coal of the Sheridan type.

PUBLICATIONS ON THE COMPOSITION OF COAL.

A limited supply of the following publications of the Bureau of Mines has been printed and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 28. Experimental work conducted in the chemical laboratory of the United States fuel-testing plant at St. Louis, Mo., January 1, 1906, to July 30, 1906, by N. W. Lord. 51 pp.

BULLETIN 85. Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913, by A. C. Fieldner, H. I. Smith, A. H. Fay, and Samuel Sanford. 1914. 444 pp., 2 figs.

BULLETIN 116. Methods of sampling delivered coal, and specifications for the purchase of coal for the Government, by G. S. Pope. 1916. 64 pp., 5 pls., 2 figs.

BULLETIN 119. Analyses of coals purchased by the Government during the fiscal years 1908-1915, by G. S. Pope. 1916. 118 pp.

TECHNICAL PAPER 2. The escape of gas from coal, by H. C. Porter and F. K. Ovitz. 1911. 14 pp., 1 fig.

TECHNICAL PAPER 5. Constituents of coal soluble in phenol, by J. C. W. Frazer and E. J. Hoffman. 1912. 20 pp., 1 pl.

TECHNICAL PAPER 8. Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 16. Deterioration and spontaneous heating of coal in storage, a preliminary report, by H. C. Porter and F. K. Ovitz. 1912. 14 pp.

TECHNICAL PAPER 35. Weathering of the Pittsburgh coal bed at the experimental mine near Bruceton, Pa., by H. C. Porter and A. C. Fieldner. 1914. 35 pp., 14 figs.

TECHNICAL PAPER 64. The determination of nitrogen in coal, a comparison of various modifications of the Kjeldahl method with the Dumas method, by A. C. Fieldner and C. A. Taylor. 1915. 25 pp., 5 figs.

TECHNICAL PAPER 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., figs 6.

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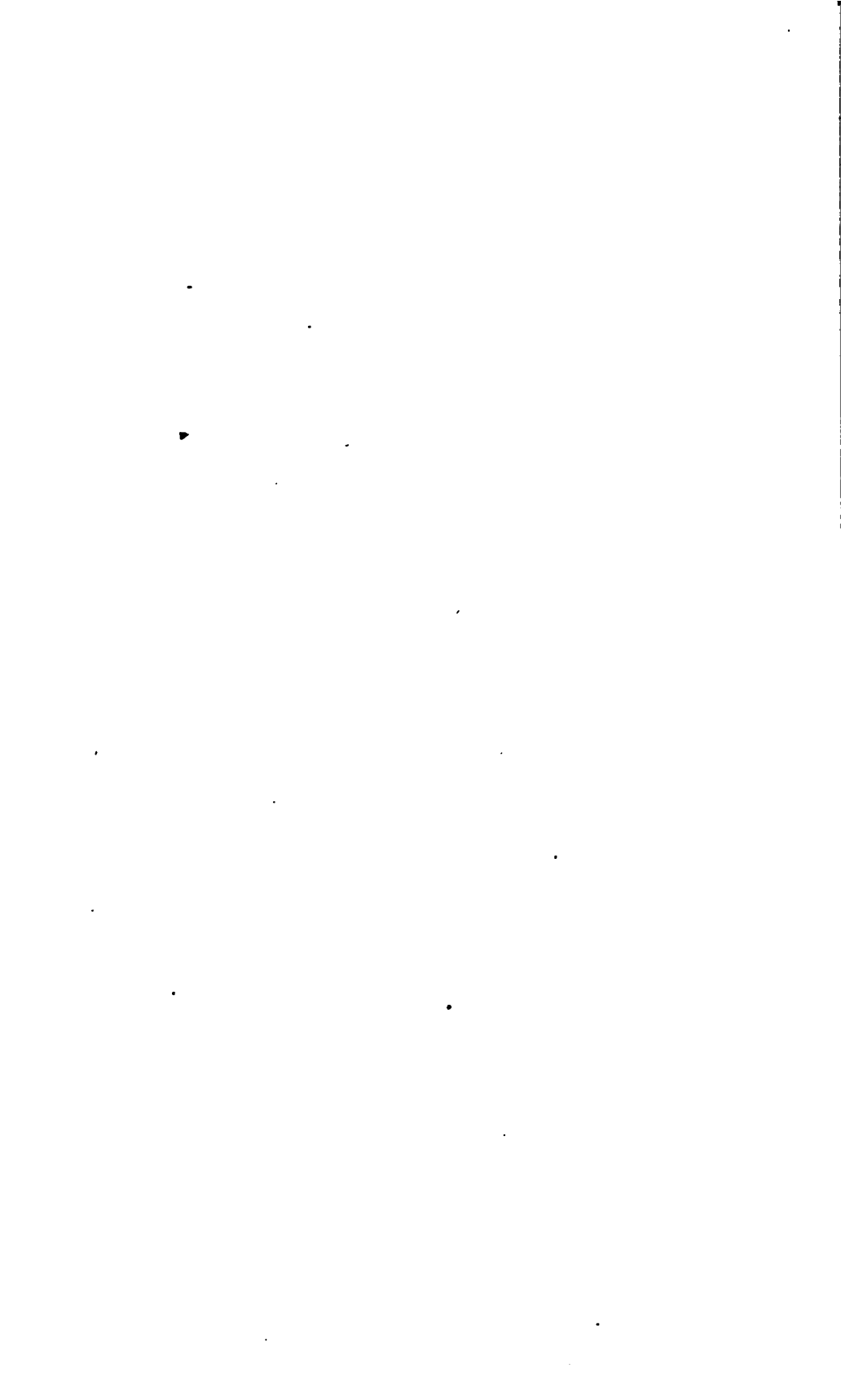
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Bulletin 137

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

THE USE OF PERMISSIBLE EXPLOSIVES IN
THE COAL MINES OF ILLINOIS

BY

JAMES R. FLEMING

AND

JOHN W. KOSTER

ILLINOIS COAL-MINING INVESTIGATIONS
COOPERATIVE AGREEMENT

[This report was prepared under a cooperative agreement
with the Illinois State Geological Survey and the engi-
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PREFACE.

In 1911 the General Assembly of the State of Illinois, with a view to safeguarding the lives of mine workers and conserving the mineral resources of the State, authorized an investigation of the mining practices and the coal resources of Illinois by the department of mining engineering of the University of Illinois and the State geological survey, in cooperation with the United States Bureau of Mines. A cooperative agreement^a was approved by the Secretary of the Interior and by representatives of the State of Illinois.

Reports of the mining practices in the eight districts into which the State was divided, a general summary of mining practices in Illinois, and a report on the general theory of subsidence have been published by the mining department of the University of Illinois. A study of the percentage of extraction of coal is now in progress.

The State geological survey has investigated and published reports on coal analyses and on the coal resources of four districts and will complete reports on the remaining districts. It has also published a report on the evidences of subsidence due to coal mining and another on the availability of roof and floor clays and shales at coal mines for the manufacture of clay products.

The Bureau of Mines has published as a result of this work: Bulletin 72, "Occurrence of Explosive Gases in Coal Mines"; Bulletin 83, "The Humidity of Mine Air"; Bulletin 99, "Mine-Ventilation Stopplings, with Especial Reference to Coal Mines in Illinois"; and Bulletin 102, "The Inflammability of Illinois Coal Dusts."

The study of mine air, mine fires, and the use of explosives is being continued, and the coking and gas-producing qualities of Illinois coals are being investigated. A detailed investigation of subsidence in Illinois is being undertaken jointly by the parties to the agreement.

VAN. H. MANNING,
Director.

^a For details of agreement see preliminary report on organization and method of investigations: Illinois Coal-Mining Investigations, Cooperative Agreement, 1913, 71 pp.



THE USE OF PERMISSIBLE EXPLOSIVES IN THE COAL MINES OF ILLINOIS.

By JAMES R. FLEMING and JOHN W. KOSTER.

INTRODUCTION.

The following report is made through the Bureau of Mines as a result of the work under the cooperative agreement with the State geological survey and the engineering experiment station of the University of Illinois. It deals with the use of permissible explosives in the coal mines of Illinois, and is intended primarily to present to mining men the possibility of assuring safer conditions in mining through the use of these explosives.

Twenty coal mines in Illinois were using these relatively safe explosives at the time this report was prepared. Of these, seventeen were in Franklin County, two were in Saline County, and one in Montgomery County. In these various fields experience had shown that although black blasting powder was a useful agent in breaking down the coal at the working face, yet it had given such conclusive evidence of its danger to life and property in causing explosions and mine fires that a substitute was sought and has been found in permissible explosives.

Each coal mine in Illinois in which permissible explosives are used was visited and a study made of the problems involved.

The report deals briefly with the dangers of black blasting powder now in such common use in coal mining. It discusses the development of permissible explosives and their introduction into Illinois coal mines and points out the characteristic differences between permissible explosives and black blasting powder. The present system of blasting and the most approved methods of using permissible explosives are described. Data are given to show that not only has safety and efficiency in these mines been greatly improved, but that the cost of explosives to the miner per ton of coal has been reduced, and that sizes of coal produced from permissible explosives compare favorably with those previously obtained at the same mines with black blasting powder. The successful use of permissible explosives in these Illinois mines as well as in many other mines throughout the country should aid in their introduction into a larger number of mines and bring about that degree of freedom from explosions and fires which should be attained in mining.

ACKNOWLEDGMENTS.

The writers desire to thank the officials of the various mining companies and the State and county inspectors for many personal courtesies and for valuable assistance in supplying the data herein contained. Especial acknowledgments are given C. M. Moderwell, president of the Illinois Coal Operators' Association, Frank Farrington, president of district No. 12, United Mine Workers of America, J. E. Jones, State mine inspector for the eleventh district, and James Dunn, superintendent of the Old Ben Coal Corporation, for suggestions and criticisms of the manuscript; to George S. Rice, chief mining engineer of the bureau, for valuable assistance given in personal supervision of the work; to Frank W. De Wolf, director of the Illinois State Geological Survey, H. H. Stoek, head of the department of mining engineering, University of Illinois, and H. I. Smith, district mining engineer of the bureau, for active cooperation; to S. P. Howell and W. C. Cope, of the explosives section of the bureau, for information on details of the work covered by their respective departments; and to J. J. Rutledge, mining engineer of the bureau, for certain field data.

THE COAL FIELDS OF ILLINOIS.

COAL RESERVES.

As reported by the Illinois State Geological Survey, the coal reserves of the State approximate 200,000,000,000 (short) tons,^a a supply sufficient to meet the entire Nation's needs for more than 200 years at the present rate of consumption, allowing for only a 50 per cent recovery of coal. There are five workable seams of commercial importance. The territory underlaid with coal embraces approximately 36,800 square miles, or 65 per cent of the area of the State.

PRESENT PRODUCTION.

During the fiscal year ended June 30, 1915,^b the mines of Illinois produced 57,601,694 tons of coal. Of this total, the No. 6 seam furnished 69.63 per cent and the No. 5 seam 20.21 per cent. These two seams, which include all the mines now using permissible explosives, therefore furnish about 90 per cent of the total output of the State.

METHODS OF MINING.

There are three general methods of mining coal in the State of Illinois, as follows:

1. Stripping.
2. Longwall advancing.

^a Preliminary report on organization and method of investigations: Illinois Coal-Mining Investigations, Cooperative Agreement, 1913, p. 25.

^b Thirty-fourth annual coal report of Illinois, State Mining Board, 1915, p. 26.

3. Room-and-pillar:

a. Common double-entry, room-and-pillar system.

b. Panel system.

In stripping operations the dangers of explosions and fires are not found as in underground workings.

In the longwall mines the roof pressure usually takes the place of explosives in breaking down the undercut face.

The room-and-pillar mines, including the common double-entry, room-and-pillar system and also the panel system, provide 94 per cent of the total output of the State. Coal undercut by machines constituted 59.1 per cent ^a of the total State production in 1915. The greater part of the remainder was from mines in which the coal was shot off the solid. In a few mines undercutting is done by hand.

AMOUNT OF EXPLOSIVES USED IN ILLINOIS MINES AND TONNAGE PRODUCED.

For the fiscal year ended June 30, 1914, a total of 60,715,795 short tons of coal was produced in Illinois. Of this amount approximately 56,400,000 tons was obtained with the use of explosives. To obtain this production, 31,607,200 pounds of black blasting powder and 930,596 pounds of permissible explosives were used. Although the amount of permissible explosives formed only 3 per cent of the total quantity of explosives used, approximately 6,000,000 tons of coal, or more than 10 per cent of the total quantity obtained by explosives, was obtained by the use of permissible explosives.^b

During the fiscal year ended June 30, 1915, of the total of 57,601,694 short tons of coal produced, approximately 53,600,000 tons was obtained with the use of explosives, 27,683,150 pounds of black blasting powder and 1,342,334 pounds of permissible explosives being used.^c In this year approximately 7,680,000 tons of coal, or 14.3 per cent of the total quantity obtained by explosives, was produced by the use of permissible explosives. The tonnage produced by permissible explosives is divided approximately as follows: Franklin County, 86 per cent; Saline County, 7 per cent; Montgomery County, 7 per cent.

Figure 1 shows the total production of coal, the amount of black blasting powder used during the period 1907-1915, and the gradual increase in the amount of permissible explosives used in Illinois mines. The figure also shows that the tonnage now produced by permissible explosives exceeds that of longwall mining. The data from which these curves were plotted, given in the table following,

^a Computed from thirty-fourth annual coal report of Illinois, State Mining Board, 1915 (p. 69), which shows that 97 mines used machines exclusively, and 34 in part, a total of 1,686 machines being used in producing 34,037,426 tons of coal.

^b Thirty-third annual coal report of Illinois, State Mining Board, 1914, p. 2.

^c Thirty-fourth annual coal report of Illinois, State Mining Board, 1915, p. 2.

18 to 30 inches above the floor; this band generally consists of bone or of gray shale.

The bed is divided into three distinct benches. The upper part, comprising about one-third of the thickness of the bed, is a brittle coal occurring often in layers 3 to 6 inches thick, separated by partings of "mother of coal." The coal in the middle part, which is separated from the upper third of the bed by a clean persistent parting of "mother of coal," is brighter and somewhat firmer and has less distinct lamination. "Sulphur" lenses are commonly found in this bench. Below the "blue band" the coal is hard and firm.

The coal in the southern part of the Franklin County field is harder than the coal in the northern part. There is no pronounced cleat and, therefore, it does not materially affect the direction of mining, as the coal may break irregularly along a number of different planes. In certain mines, however, there is a slight vertical cleat in a northeast and southwest direction, which permits freer shooting of the coal on a face parallel with this direction. Although the cleat of the coal has little effect on the direction of mining, the joints in the roof material have; for frequently when work is driven north or south the roof tends to break so much as to increase mining costs considerably.

The "blue band," which is characteristic of the No. 6 coal, largely determines the placing of drill holes in blasting. After the coal is undercut by machines, it is "snubbed"; that is, a sloping cut extending from the "blue band" to about half the depth of the undercutting is made. When "snubbing" shots are fired or where "snubbing" is done by hand, this parting serves as a guide to the height to which the coal is to be "snubbed."

The roof is commonly of shale, which is 15 to 110 feet thick, but in some of the latest mines east of Benton, limestone is found immediately above the coal. The shale breaks when exposed, and for this reason the top layers of coal are left in place to form the roof. Usually 10 to 18 inches and at times as much as 40 inches is left up. In some instances the top coal is recovered after the rooms are mined out. The floor is clay.

Plate I^a shows the No. 6 coal bed and the overlying shale exposed, the top bench of coal (dark part) which is left up in mining, and the two lower benches separated by the "blue band," which occurs about 2 feet from the bottom. The structure of the lower 5 feet of the No. 6 coal bed, including the blue band, is shown in Plate II, A.

In Montgomery County, the characteristic "blue band" is present. The coal is hard and firm and averages about 7½ feet thick. The roof is shale and limestone. Where light-colored shale is present

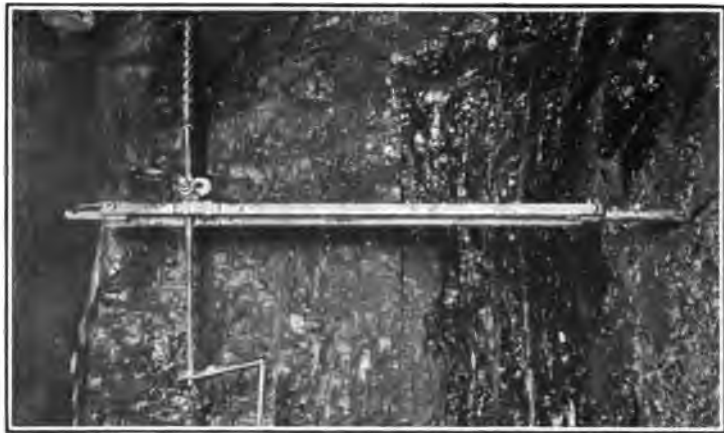
^a Cady, G. H., Coal resources of District 6: Illinois Coal-Mining Investigations, State Geol. Survey Bull. 15, 1916, p. 92.



BROKEN ROOF IN ENTRY OF AN ILLINOIS MINE. CONDITIONS TYPICAL OF NO. 6 BED WHEN THE SHALE OVERLYING THE COAL BREAKS DOWN.



A. LOWER 5 FEET (TWO BOTTOM BENCHES) OF NO. 6 COAL IN FRANKLIN COUNTY. BLUE BAND JUST BELOW CENTER OF PICTURE.



B. POST DRILL, USED IN ILLINOIS COAL MINES.



C. GRIP DRILL, USED IN ILLINOIS COAL MINES.

it forms a good roof. Where the roof is dark shale the top bench of coal, 15 to 18 inches thick, is left in place. The coal is underlain with a bed of clay 4 to 50 inches thick.

The coal as mined from these beds is uniformly hard and firm and withstands shipment well, especially in the smaller sizes, and is well adapted for domestic or steam fuel.

OCCURRENCE OF GAS IN THE COAL BEDS.

In Franklin County, coal bed No. 6 lies at a depth ranging from 417 to 700 feet.^a The part of the southern Illinois coal field east of the Duquoin anticline, including Franklin, Williamson, and Saline Counties, is the most gaseous in Illinois. Although on the whole the mines are not regarded as being gaseous enough to require the use of locked safety lamps, yet considerable methane is given off from the coal. Numerous local faults occur in both No. 5 and No. 6 seams, frequently giving off local feeders of gas and creating the possibility of dangerous accumulations. Darton^b found that the principal mines in Franklin County in 1912 generated on an average about 200 cubic feet of methane per minute under normal working conditions. In the more gaseous areas, immediately above the coal there is a considerable thickness of impervious shale, whereas in the less gaseous districts in which the same seams are mined, jointed limestone closely overlies the coal.

The results of laboratory tests^c indicate that the dust from the No. 6 coal bed is highly inflammable as compared with other Illinois coal dusts and when raised in a cloud might be ignited by a blown-out shot of black blasting powder. Considerable fine rib dust is nearly always in evidence in the mines in the No. 6 bed. As the top coal is left in place, and shale from the roof does not fall on the roadways, and as haulage is by electricity, so that the bottom clay is not ground up much by the feet of men and animals, the road dust consists almost wholly of coal dust.

CAUSES LEADING TO THE USE OF PERMISSIBLE EXPLOSIVES IN ILLINOIS.

DEVELOPMENT OF THE FRANKLIN COUNTY COAL FIELD.

In the Franklin County field, where the use of permissible explosives in Illinois was begun, mining is comparatively recent. The field was opened at Zeigler in 1903. Mines were developed at West Frankfort, Benton, Christopher, and other points in Franklin County, until at the present time this field ranks as one of the important bituminous coal areas in the United States. New mines are being developed.

^a Average depth at shaft for mines listed is 541 feet, see Table 1, p. 22.

^b Darton, N. H., Occurrence of explosive gases in coal mines: Bull. 72, Bureau of Mines, 1915, p. 219.

^c Clement, J. K., Scholl, L. A., The inflammability of Illinois coal dust. Bull. 102, Bureau of Mines.

The Franklin County mines in which permissible explosives are used have an average daily production of approximately 2,800 tons. Six of these mines have a daily capacity of over 4,000 tons each. Of the total of 57,601,694 short tons of coal produced in Illinois for the year ended June 30, 1915, 7,324,644 tons was produced in this county, of which about 90 per cent was shot down with permissible explosives.

EXPERIENCE IN FRANKLIN COUNTY WITH BLACK BLASTING POWDER.

The first mines in Franklin County were opened when black blasting powder was the only practical coal-mine explosive used in the United States. From the beginning much trouble was experienced with mine fires caused by its use. The gaseous conditions in this field, the extremely inflammable nature of the coal dust, the methods of mining, by which 10 to 40 inches of coal left for roof spalled onto the roadways and produced dust, all combined to make the use of black blasting powder dangerous and costly as regards life and property.

A number of destructive explosions happened in Franklin County in 1908 and 1909. Such occurrences were not confined to any particular mine, although certain mines had more than one, but were scattered over the entire field. Furthermore, every mine in the district suffered from frequent fires from the use of black blasting powder, the coal igniting readily and there being some methane liberated at the face. At some mines as many as twenty fires occurred nightly, following the firing of shots, and in two instances 40 were reported. The employment of fire runners to follow the shot firers and extinguish fires became necessary. Four to ten men were employed for this work at each of the various mines, and the relatively small number of fires getting beyond control speak well of the efficiency and carefulness of these men and of the officials.

At one mine in this district two destructive coal-dust explosions, primarily caused by firing shots of black blasting powder, occurred in an interval of about six weeks, the first on November 5, 1908, and the second on December 12, 1908. Immediately after the first explosion the governor of the State ordered that a thorough examination of conditions in this mine be made by the State inspectors of mines and the State mining board. That the explosion was initiated by a shot and was accompanied with much flame and violence is pointed out in the following extract from the report of this committee.^a

Several tests with safety lamps were made on the roadways, at the face of the working places, and particularly at the high points where falls of "slate" had occurred, but no evidence of gas was discovered. After reaching the face of the first northeast entry

^a Twenty-eighth annual coal report of Illinois, State Bureau of Labor Statistics, 1909, p. 372.

beginning with room No. 20, we proceeded down until No. 8 was reached. As we approached that point, indications of serious disturbances increased—falls of "slate" and coal crushed from pillars, broken timbers, mining machines, and pit cars completely destroyed. From the mouth of room No. 8 and continuing up to the face thereof, evidences of intense heat and fire on the roof and props were observable. At the face of that room we found that several shots had been fired on the night of the explosion; in fact, they were the only shots fired in any of the rooms in that entry that evening. It furthermore appeared that the shots had been placed near the top of the coal, and as the seam of coal is much softer near the roof much of it that was blown down was in a badly shattered condition. There is no question but that the direct cause of the explosion originated from these shots in the face of room No. 9, first northeast entry; that the placing of these shots so near the roof and the broken condition of the coal indicated an overcharge of powder. * * * In order to avoid a repetition of such disasters, we recommend that in all cases where coal is undercut with chain machines, the coal be snubbed or blocked down at not more than 3, or less than 2, feet from the bottom of the seam, and that all undercuttings produced by said machines be collected and loaded out before any shots are fired.

An explosion at shot-firing time also occurred shortly thereafter at a mine near Benton and another in a mine near West Frankfort. Fortunately, large loss of life was prevented, as the miners were out of the mine at the time, there being a State law requiring that all men except shot firers should be out of the mine during shot firing.

Data as to the costs of these destructive explosions throughout the field are not obtainable, but a heavy expense must be charged against explosions and fires from the use of black blasting powder. In one of the accidents mentioned above, the mine was practically new, and had shipped only about 150,000 tons of coal, when an explosion occurred at shot-firing time, which killed the shot firers and set the mine on fire, the flames reaching the top of the hoisting shaft and preventing any recovery operations. Both shafts had to be sealed. The shafts were kept sealed for 120 days and later the mine was flooded.^a Shipments of coal were suspended for a period of over a year while the mine was being restored. The total loss due to flooding, suspension, and restoring the mine to working condition was upward of \$175,000.

AGREEMENT BETWEEN FRANKLIN COUNTY OPERATORS AND MINERS.

A feeling of extreme uneasiness existed at this time, and operators and miners considered means that might end such disasters. This crystallized in the agreement between Franklin County operators and miners, which commendably aimed to improve the methods of using black blasting powder. As the recommendations contained in this agreement were a step forward and illuminate some special features of preparing the coal face preliminary to shooting, they are given below. "Snubbing" of the coal, the loading out of machine cuttings

^a Twenty-eighth Annual Coal Report of Illinois, State Bureau of Labor Statistics, 1909, p. 375.

prior to blasting, tamping the shot holes with clay and not with coal dust or cuttings, prescribing that the drill holes be not nearer than 12 inches to the ribs of the working place, and shorter by 10 inches than the depth of the undercutting are all good rules to-day for use in shooting with permissible explosives in this field. The agreement was as follows:

AGREEMENT RELATIVE TO PREPARATION OF FACE FOR BLASTING IN FRANKLIN COUNTY MINES.*

1. That the bits used in the Franklin County chain-machine mines be the maximum width of 2½ inches, on a 6-foot bit, and the length not more than 6 feet.
2. That a minimum of 25° and a maximum of 45° holes across the face be drilled for the "buster" shots, and no more than two holes be fired in all wide working places in one shift, unless more holes are required to furnish coal for the day's work.
3. We recommend that all coal be "snubbed" with pick and wedge.
4. That "snubbing" means that the coal must be snubbed not lower than the height of the blue band and back one-half the depth of the cutting, and all coal snubbings must be removed from underneath.
5. That all cuttings from the electric chain machine be loaded out before any shots are fired in the working places, the company to furnish cars promptly for loading same.
6. That all drill holes be tamped with clay, the company to furnish clay when it is impossible for the miner to obtain the clay within a distance of 300 feet of his working place.
7. That all shots, when shot by shot firers, shall be inspected before being tamped, and a vigilant watch on all shots be kept by the shot firers, and any excessive use of powder used in any holes by the miner and any dangerous holes be reported to the mine manager, and said miner shall be discharged.
8. That all holes be tamped on a copper needle, unless holes be wet, and we recommend that no shots be fired with fuse.
9. That no holes be drilled until after the working places are cut, only in emergency cases, then "buster" shots may be drilled, and all rib holes must be drilled so the powder will lay no closer than 12 inches of the rib of the working place, and all holes must be drilled at least 10 inches less than the depth of the cutting, thus giving the powder a chance to work.

SEPARATE CLAUSE.

In case the "snubbing" is done with powder, it is understood that the snubbing holes shall not be more than 2 feet 10 inches above the top of the cutting, and that said snubbing shall extend the entire width of the place and shall be loaded out to within 2 feet of the back of the cut, provided the snubbing of coal by pick and wedge can not be agreed upon by the joint boards.

NEED OF A SUBSTITUTE FOR BLACK BLASTING POWDER.

The spirit of the foregoing agreement indicates an earnest desire on the part of the mining fraternity to cope with the problems presented. The results, although showing an improvement over previous conditions, were unsatisfactory. The relief was partial. Fires continued to occur at shot-firing time, and the explosion in the mine at Benton when solid shooting was in vogue was repeated later after

* Adopted in January, 1906.

mining machines were installed. A violent explosion occurred at one mine from a shot which was apparently properly placed with less than two pounds of powder in undercut coal. It was seen that although black blasting powder was an effective agent in the production of coal, yet its characteristic long hot flame rendered it unsafe in these mines, even when used by expert miners under stringent regulations.

The problem of mine fires and explosions has not been confined to Illinois. In all bituminous coal mines, especially where explosive gas and dust have been present, the need of a substitute for black blasting powder has been evident. As early as 1854 the problem was demanding attention. In that year the London Mining Journal referred to the testimony of Mr. George Elliot, a witness examined by the Parliamentary Committee,^a as follows:

"A very painful accident happened at Usworth Colliery, of which I was owner and manager, some time since, and which arose from fire of shot; and since that Mr. Lee Patinson, a practical chemist, and myself have been endeavoring to ascertain whether we could invent some power to bring coal down without an explosive mixture, such as gunpowder; and I am sorry to say that no very successful result has yet been arrived at; but up to the present time, I am afraid, notwithstanding gunpowder is a very old chemical invention, that very little progress has been made since it was first used and that we are in ignorance of any substitute for it of equal power. If public attention were called to it, perhaps science might discover some substitute."

And as this journal had previously solicited public notice as to the subject, we again repeat that the researches of chemical and electrical discovery, which have in our times produced such marvelous results, could not be devoted to a nobler object of scientific ambition.

Although statistics of the past 27 years show that only 15.4 per cent of all fatal accidents in the coal mines of Illinois^b have been caused by explosives or by explosions of gas or coal dust due to firing of shots or other initiating cause, yet mine explosions are nevertheless dreaded because of their far-reaching effects. The conditions that exist in coal mines tend to make explosions a menace to the lives of everyone in the mine, and therefore any factor or combination of factors that might form the initiating causes should be carefully controlled. The most careful man in the mine may meet his death through no fault of his own, but through the carelessness or thoughtlessness of a fellow workman. Miner and operator are alike affected by the dangers of using long-flame explosives such as black blasting powder, in gaseous and dusty mines.

Experiments during the past few years have conclusively proved that:

1. The flames of black blasting powder and of dynamite readily ignite an inflammable mixture of methane and air.

^a Report of the Parliamentary Committee on Accidents in Mines, London, 1854.

^b Editorial, London Mining Journal, 1854, vol. 24, p. 315.

^c Thirty-third annual coal report of Illinois State mining board, 1914, p. 106.

2. An explosion of coal dust can be initiated by a single blown-out shot from black blasting powder at the face.^a

3. The amount of coal dust, suspended in the air, that may propagate an explosion may be very small. Taffanel, in France, obtained explosions regularly with 190-mesh bituminous dust with a weight of 0.07 ounce per cubic foot of dust cloud. In experiments in the gas and dust gallery No. 1 at the Pittsburgh station of the Bureau of Mines, propagations have been obtained with 200-mesh dust from the Pittsburgh coal seam with as low a weight as 0.032 ounce per cubic foot of gallery space.^b

4. As little as 1 to 1½ per cent of methane,^c an amount that may often be present without being detected with the safety lamp, will, as has been shown by tests of the Bureau of Mines, greatly increase the explosibility of coal dust.

With these facts available, it should be apparent that the use of black blasting powder and dynamite will always be an element of danger where gas and coal dust are present. The concussions from the shots in the various parts of the mine cause the supposedly harmless road and rib dust to become suspended in the air, and then a blown-out shot, or even an overcharged or undercharged shot, may initiate an explosion. It may be possible to estimate the number of actual explosions that have occurred in a particular mining district, but there are no means of determining how many explosions were prevented only by the absence of only one contributory factor or by the presence of some hindering influence. Freedom from explosions in certain mines in which black blasting powder is used, where conditions are known to border on the dangerous, is not because of systems in vogue, but in spite of them.

There is no absolute guaranty against mine explosions from black blasting powder happening even in nongaseous mines. In addition to data obtained from tests at the experimental mine of the bureau there is other proof. For example, an explosion occurred at shot-firing time in a mine in the Springfield district, Ill., in March, 1915, which killed two shot firers, although no previous trouble had been experienced in 17 years of operation. Analyses of return air

^a The standard method of initiating an explosion at the Bureau of Mines experimental mine near Pittsburgh, Pa., is by firing from a cannon a charge of 4 pounds of FFF black powder stemmed with dry clay producing a blown-out shot, in the presence of finely ground coal dust distributed in quantities of 2 pounds per linear foot. This quantity (4 pounds) of black powder is used to insure strong initial explosion of the coal dust, but in some experiments only 2 pounds has been used, placed in a drill hole in the face in solid coal, the single shot causing a violent dust explosion. Coal-dust explosions have been caused by black powder shots when the loading of coal dust was only 0.3 pound per linear foot of entry, which is equivalent to only 0.083 ounce per cubic foot of space. This test was made with finely ground coal dust. Coal dust that is all fine impalpable dust is most highly explosive and behaves much like an inflammable gas when raised in a cloud in the air and ignited. See Rice, G. S., and others, *First series of coal-dust explosion tests in the experimental mine*, Bull. 56, Bureau of Mines, 1913, 113 pp.

^b Rice, G. S., *Explosibility of coal dust*, Bull. 20, Bureau of Mines, 1911, pp. 101-102.

^c Rice, G. S., and Jones, L. M., *Methods of preventing and limiting explosions in coal mines*: Tech. Paper 84, Bureau of Mines, 1915, p. 7.

at this mine show that methane is not present in appreciable percentages. A coal-dust explosion happened at a mine in the Peoria district, in February, 1915, during the firing of shots of black blasting powder by the night shift. Two entrymen who had lighted the shots and were on their way out of the mine were killed.

DEVELOPMENT OF SHORT-FLAME OR "PERMISSIBLE" EXPLOSIVES.

The first steps in the manufacture of permissible explosives date back about 30 years. As stated by Rice,^a research was begun in Germany in 1885 for the development of explosives "Which would not ignite fire damp and coal dust under ordinary conditions," and in the coal-mining countries of Europe, including England, France, Germany, Belgium, and Austria, it is now required that only such explosives shall be used in coal mines as have passed certain tests or have met certain requirements.

So-called "safety" blasting powders were introduced into the bituminous coal fields of Pennsylvania and West Virginia in 1902, but it was not until 1908, soon after the establishment of the Pittsburgh testing station of the Bureau of Mines where testing of explosives^b for use in gaseous and dusty mines was begun, that an impetus was given to the manufacture and use of such explosives in the United States. Since that time constant effort has been made by manufacturers to improve the qualities of their products, with the result that many of the early brands have been withdrawn from the market and their places taken by better ones. The February, 1917, list of explosives contains the brand names of 153 "permissible" explosives.^c

WHAT IS A PERMISSIBLE EXPLOSIVE?

It should be noted that an explosive is called a "permissible explosive" when it is similar in all respects to the sample that passed certain tests^d by the Bureau of Mines, and when it is used under the following conditions:

1. That the explosive is in all respects similar to the sample submitted by the manufacturers for the test.
2. That detonators—preferably electric detonators—are used of not less efficiency than those prescribed, namely, those consisting by weight of 90 parts of mercury fulminate and 10 parts of potassium chlorate (or their equivalents).

^a Rice, George S., The explosibility of coal dust: Bull. 20, Bureau of Mines, 1911, pp. 82-83.

^b Systematic tests of permissible explosives began early in 1909 at the experiment station at Pittsburgh. The first list of explosives passing the tests was published by the bureau May 15, 1909.

^c Fay, A. H., Coal-mine fatalities in the United States during February, 1917, and list of permissible explosives, lamps, and motors tested prior to March 31, 1917, 24 pp.

^d For details of character of tests see pp. 101, 102 of this report, and Schedule 1, Fees for testing explosives and conditions and requirements under which explosives are tested, Bureau of Mines, 1913, 8 pp. For a detailed description of the methods used see Hall, Clarence, and Howell, S. P., Tests of permissible explosives, Bull. 66, 1913, 315 pp.; Storm, C. G., The analysis of permissible explosives, Bull. 96, 1914, 86 pp.

3. That the explosive, if frozen, shall be thoroughly thawed in safe and suitable manner before use.

4. That the quantity used for a shot does not exceed $1\frac{1}{2}$ pounds (680 grams), and that it is properly tamped with clay or other non-combustible stemming.

The general subject of permissible explosives has been treated in various publications ^a of the Bureau of Mines.

Further explanation from the chemist's point of view as to what constitutes a permissible explosive is given in the following statement by C. G. Storm,^b former explosives chemist of the Bureau of Mines:

Permissible explosives are blasting explosives that, having passed certain tests prescribed by the Bureau of Mines, are considered suitable for use in gaseous or dusty coal mines when used in the manner prescribed by the bureau. The chief characteristics that an explosive must possess in order to pass the tests are: (1) Relatively low flame temperature on explosion, (2) a minimum amount of explosion flame. Mixtures of natural gas and air are exploded when exposed to a temperature of about 650° C. for about one-tenth of a second, but are not acted upon by flame of much higher temperature if the flame is of sufficiently short duration. Thus, although the flame temperature of most permissible explosives lies between 1,500° and 2,000° C., they do not ignite the most explosive mixtures of gas and air because of the rapidity of their explosion, the duration of their flame being only about two to five ten-thousandths of a second. Black blasting powder, on the other hand, has a flame temperature of over 2,200° C., and a duration of flame, under the same conditions of test, of approximately one second. Ordinary 40 per cent nitroglycerin dynamite gives a flame duration of only about four ten-thousandths of a second, but its flame temperature is about 2,900° C. Both black blasting powder and dynamite, then, fail to pass the tests for permissibility, black blasting powder because of the long duration of its flame and dynamite because of the high temperature of its flame.

The problem in producing permissible explosives is therefore to formulate explosive mixtures that, while giving a minimum amount of flame of short duration and low temperature, will develop sufficient energy to do the work of breaking down coal in an economical manner. An example is cited below:

By proper additions of various ingredients, ordinary dynamite can be so altered in composition that its flame temperature will be reduced sufficiently to render the resulting explosive "permissible." If, for example, a large excess of carbonaceous combustible material, such as wood pulp, flour, or corn meal, is added, the gaseous products of explosion are so altered that the flame temperature is greatly reduced. The same effect is obtained by the addition of water, either in the liquid state, in which form it is absorbed by the other ingredients of the explosive, or in the form of water of crystallization in such crystalline salts as alum, or magnesium sulphate, both of which con-

^a Rutledge, J. J., and Hall, Clarence, The use of permissible explosives, Bull. 10, 1912, 34 pp. Hall, Clarence, Snelling, W. O., and Howell, S. P., Investigations of explosives used in coal mines, 1911, 197 pp. Munroe, C. E., and Hall, Clarence, A primer on explosives for coal miners, Bull. 17, 1911, 61 pp. Hall, Clarence, and Howell, S. P., Tests of permissible explosives, Bull. 66, 1913, 313 pp. Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen, Bull. 80, 1915, 125 pp. Storm, C. G., Tests of permissible explosives, Bull. 96, 1916, 88 pp. Watteyne, Victor, Malsamer, Carl, and Desborough, Arthur, The prevention of mine explosions, Tech. Paper 21, p. 7. Fay, A. H., Production of explosives in the United States during the calendar year 1912, Tech. Paper 69, 1914, 8 pp. Fay, A. H., Production of explosives in the United States during the calendar year 1913, Tech. Paper 85, 1914, 15 pp. Howell, S. P., Permissible explosives tested prior to Mar. 1, 1915, 16 pp. Fay, A. H., Production of explosives in the United States during the calendar year 1914, with notes on coal-mine accidents due to explosions, 1915, 16 pp. Fay, A. H., Monthly statement of coal-mine fatalities in the United States.

^b Storm, C. G., In communication to the Bureau of Mines, July, 1914.

tain about 50 per cent of water of crystallization. Similarly, the addition of inert solid matter, such as clay or powdered rock, which simply absorbs part of the heat liberated by the explosive reaction, or of readily volatile inert materials such as certain ammonium salts, which consume heat in being volatilized, will also produce the characteristics of permissible explosives.

Although such additions naturally lower the strength of the explosive, it is possible to obtain the desired end and still produce explosives entirely suitable for use in coal mining, as is evidenced by the fact that there are at present (July, 1914) about 125 explosives on the "permissible list."

Careful chemical analyses are made of every explosive received for test, in order to determine whether the explosive possesses any objectionable chemical features. Field samples, collected from time to time, are also analyzed in order to ascertain whether the different permissible explosives are being manufactured in accordance with the composition of the original sample submitted for test.

The difficulties encountered by the chemist in analyzing such explosives may be realized when it is known that the manufacturers use a great variety of ingredients in bringing about the desired results. About 60 to 70 ingredients have been found in the various permissible explosives analyzed in the bureau laboratory.

This short duration of flame may be more clearly expressed to the man at the working face by two simple illustrations:

The finger of one's hand after moistening may be touched against a red hot stove or iron without any sensible effects, if the removal action is sufficiently quick. The flame of a burning match may be rapidly passed over a grain of powder or a match head without causing ignition. In either instance, the flame temperature is sufficient to cause burning or ignition, but the time of contact is so short that no effect is produced.

CLASSES OF PERMISSIBLE EXPLOSIVES.

The four general classes of permissible explosives ^a and the characteristic materials of composition are as follows:

Classes of permissible explosives and characteristic material.

Class.	Characteristic material.
Class 1. Ammonium nitrate explosives.	Ammonium nitrate.
Class 2. Hydrated explosives.....	Nitroglycerin with salts containing water of crystallization.
Class 3. Organic nitrate explosives.....	Organic nitrates other than nitroglycerin.
Class 4. Nitroglycerin explosives.....	Nitroglycerin.

Of these four types, nitroglycerin explosives are most commonly used in Illinois mines. Nitroglycerin explosives contain free water or an excess of carbonaceous combustible material, which is added to reduce the flame temperature. A few explosives of this class contain salts that reduce the strength and shattering effect of the explosives on detonation. Nitroglycerin explosives have the advantages of detonating easily and of not being readily affected by moisture.

^a For detailed description of classes of permissible explosives, see Howell, S. P., *Permissible explosives tested prior to March 1, 1915*: Tech. Paper 100, Bureau of Mines, 1915, 16 pp.

RATE OF DETONATION.

The rate of detonation may be defined as the rate of explosion, or the rate at which detonation travels through a given length of explosive. When an explosive, which may consist of a solid or liquid substance, or a mixture of substances, is acted upon by a blow or by an initiating compound, such as mercury fulminate, more stable substances, largely or entirely gaseous, are formed almost instantaneously. This conversion process is called detonation, and the rate of detonation is the rate at which the conversion takes place through a given length of explosive. Detonation is more rapid than the transmission of flame from particle to particle, as is the case with black blasting powder.

The rate of detonation is important in relation to the use of permissible explosives in coal mining. It is thought that explosives having an intermediate rate of detonation are best suited for shooting average bituminous coals when undercut. Hall and Howell ^a state that—

The rate of detonation is the governing factor in judging the efficiency of an explosive, and it offers the best single means for selecting explosives suitable to meet the varying conditions of coal mining.

To meet the varying conditions of coal mining in this country, the explosive manufacturers have devised explosives with rates of detonation that range from 4,750 to 14,560 feet per second.

CHARACTERISTIC ACTION OF EXPLOSIVES.

There is a marked difference in the characteristic action of black blasting powder and that of permissible explosives and other detonating explosives. An understanding of this difference is of assistance in the proper use of permissible explosives.

When laid on top of a rock and exploded, black blasting powder does not materially affect the rock, because it explodes so slowly that the gases formed can lift the air above them and escape; whereas dynamite, if laid on brittle or soft rock and detonated, may shatter it, because the dynamite explodes so quickly that the great volume of gases formed can not lift at once the air confining it and must exert pressure on the rock. This action has given the erroneous impression that dynamite and likewise permissible explosives act downward, rather than upward and outward. Experiments show that the force exerted by an explosive is equal in all directions.^b

On account of the relatively quick action of permissible explosives, it has also been supposed that no stemming, or very little stemming, is necessary when they are used for shooting down coal. Actual tests have proved that the use of efficient stemming may increase the useful

^a Hall, Clarence, and Howell, S. P., Tests of permissible explosives: Bull. 66, Bureau of Mines, 1913, p. 20.

^b Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen: Bull. 80, Bureau of Mines, 1915, p. 14.

energy of a shot 60 to 93 per cent.^a Some miners have claimed that merely plugging the mouth of the drill hole with a lump of clay is a desirable method. Such practices should not be permitted. Properly confining an explosive is the cheapest and best way of using it so as to obtain the most efficient results.

The rate of burning of black blasting powder, size FF, when confined is approximately 1,540 feet per second.^b The larger sizes as used in Illinois mines, principally F and C, have a still slower rate of burning, whereas the ordinary grades of permissible explosives detonate at a rate of about 7,500 feet per second when unconfined and at a higher rate when confined. Although a charge of black blasting powder in a drill hole may start to burn at a uniform rate when ignited, the rate of burning as the coal breaks and cracks form and the stemming is blown or shoved back may decrease before the charge is completely burned, as the powder tends to burn slower when the pressure is released. When a drill hole contains more black blasting powder than is necessary to break the coal free, the coal falls before all the powder has burned and the burning powder may ignite any gas liberated or any dust raised by the shot.

The same situation would arise if a hole was drilled to within a few inches of the face of another room or working, as in "holing through" a crosscut, or in slabbing pillar coal; the coal beyond the hole would offer little or no resistance to the blast. Often a poorly tamped hole results in a blown-out shot, and the flame of the burning powder may extend several feet from the mouth of the drill hole, and if inflammable dust or a small percentage of inflammable gas is present may initiate an explosion. Even when the shot appears to be correctly charged, its effect may be that of an overcharge in the presence of a slip, a fault, partings, or soft streaks in the coal, or of an undercharge in the presence of thick "sulphur" bands, with the result that the shot "throws" flame.

One of the fundamental distinctions between black blasting powder and permissible explosives is that black blasting powder by reason of its comparatively slow rate of burning exerts a wedging or heaving action, and as pointed out, any excess energy over that required to break down the coal may be expended on the air, possibly giving rise to a "windy" and therefore dangerous shot; whereas with a permissible explosive the action is more instantaneous and any excess energy of the explosive is expended on the coal with some shattering effect, but with the danger of a blown-out shot reduced to

^a Snelling, Walter O., Hall, Clarence, The effect of stemming on the efficiency of explosives: T. P. 17, Bureau of Mines, 1912, p. 18.

^b Hall, Clarence, Snelling, W. D., and Howell, S. P., Investigations of explosives used in coal mines with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe: Bull. 15, Bureau of Mines, 1911, p. 174.

a minimum if the charge limit has not been exceeded. Because of the quick action of permissible explosives the amount used should be judged closely. The effect of the sudden release of pressure from the discharge of a permissible explosive is often observed on the stemming of a well-tamped shot. The clay plug will be compressed and hardened for several inches at the end that was nearest the explosive, but the rest of the plug will not be affected.

However, the shattering effect of a permissible explosive in a well-directed and properly charged hole is confined to the coal in the immediate vicinity of the bore hole, and where the coal is undercut the amount of lump coal obtained is equal to that obtained with black blasting powder.

GROWTH IN THE USE OF PERMISSIBLE EXPLOSIVES IN ILLINOIS MINES.

The use of permissible explosives in Illinois, although covering only a comparatively recent time and restricted areas, nevertheless forms an interesting part of the coal-mining history of the State. Practically all early mining with explosives was done with black blasting powder; therefore miners were accustomed to its use, and even though acquainted with its dangers in a general way, they accepted these as a necessary hazard of the industry. After the explosions recorded in 1908, relief was sought. About this time, the Federal Government had through experiments brought to the attention of the mining public the fact not generally acknowledged that a blown-out shot of black blasting powder could cause a violent explosion with certain coal dusts without the presence of inflammable gas, and had advanced the possibilities of increased safety through the use of permissible explosives. Illinois coal operators began to investigate the merits of these short-flame explosives.

Perhaps the first tests in the State of permissible explosives were made at Dering No. 18 mine, West Frankfort, in 1909. The results were favorable. Shortly thereafter tests were made in various mines to determine the success or failure of permissible explosives under existing conditions.

At the United Coal Co.'s No. 1 mine, near Christopher, trial shots were made, and it was at this mine that later the tests for determining the prices at which permissible explosives should be supplied to the miners, under the working agreement between the operators and the miners, were conducted. At the first test five shots were arranged by the explosives manufacturer's representative, and it was then decided that the shots be fired by the shot firers in the usual manner. The officials of the coal company were present and waited to go below to observe the results. After the shots were fired, a telephone message

came from below to the effect that the shots had not worked. This, of course, was disappointing, and the officials returned to headquarters thinking the results a failure. When the coal was loaded out the following day it was seen, however, that four of the five shots had done good work, and that the shot firers had arrived at their conclusions, not by a careful examination, but more from the detonations of the blasts, the report being somewhat sharper than the well-known resounding "boom" of black blasting powder. More tests were made with favorable results, and in a short time about 75 per cent of the miners in this mine were using permissible explosives, although some of the miners were still opposed to their use. Fortunately there were a number of miners who had used similar explosives in England, Wales,^a and Belgium, and these were in a receptive mood, but to the majority of miners the use of these explosives was new.

Tests were also made at the Hart-Williams No. 1, Benton No. 1, Zeigler District No. 1, Christopher, Rend, Sesser, Zeigler, Dering No. 11, and Hanaford mines, and the use of permissible explosives was slowly introduced in these mines. In a number of mines the miners were given their choice of using either a permissible explosive or black blasting powder. In two mines this resulted in discontinuing the use of black blasting powder through popular choice. In another mine, where for a year permissible explosives were used in entries and black blasting powder in rooms, fires continued to occur regularly in rooms, but none in entries, and finally permissible explosives came to be used entirely. It was found that where the coal was properly undercut and "snubbed" the use of permissible explosives gave as good results as black blasting powder and little, if any, increase in the production of slack coal.

In 1910 two new mines were opened, the United Coal Co. No. 2 mine at Buckner and Old Ben No. 8 mine at West Frankfort. These mines, each with a daily capacity of 4,000 tons, have used only permissible explosives from the beginning.

By 1911 several mines were using permissible explosives exclusively. The West mine at West Frankfort, opened in 1912, began their use in 1913. Since 1913 all the mines opened in this district have used permissible explosives, including Old Ben No. 9, Christopher No. 2, and the Orient mine. Old Ben No. 9 mine was developed entirely with permissible explosives, producing more than 2,000 tons daily within 11 months from the time the coal was reached.

The Shoal Creek Coal Co., at its Panama mine, having had much trouble from local explosions and fires from use of black blasting powder, made a change to permissible explosives in the spring of 1913,

^a In Great Britain those explosives which pass the required tests are called "permitted explosives."

a special test being conducted by a local powder commission.^a In July, 1914, the Saline County Coal Co. made a change from black powder without shot firers to permissible explosives with shot firers at the Saline No. 3 mine, Harrisburg. The joint State powder commission^a issued the following statement, June 5, 1914, in settlement of the powder question at this mine:^b

We, the joint State powder commission, who were called to review a local test held by the local powder commission at Saline County Coal Co.'s mine No. 3, find that the test was carried out as per contract. We, therefore, jointly recommend that when "permissible explosive" is installed in this mine in order to insure greater safety of the miners and to protect the company from mine fires the brand agreed upon by the local powder commission shall be the brand of permissible explosive used during the life of this contract, unless otherwise mutually agreed upon. Detonators will be delivered at the shaft bottom and permissible explosive be delivered as per State contract.

In January, 1916, the New mine of the Middle Fork Mining Co. was first operated on a commercial basis; permissible explosives are being used exclusively.

The Eldorado Coal & Mining Co. began to use permissible explosives in April, 1916, at the Seagraves mine, Eldorado, after a month's test. During a period of 14 months 5,372 fires were reported at this mine from black blasting powder, and the introduction of permissible explosives fills a long-felt need.

The results of the use of permissible explosives have been most favorable in these various mines. Data as to costs of explosive per ton, sizes of coal produced, and the advantages of permissible explosives are discussed later in this report.

DATA ON MINES USING PERMISSIBLE EXPLOSIVES.

Figure 2 shows the location of Illinois mines in which permissible explosives are now being used. Table 1 lists the companies using permissible explosives, gives data as to the location of mines, date of opening of mine, date of starting to use permissible explosives, material of construction of surface magazine, approximate average daily tonnage, depth of coal from surface, average thickness of coal, total number of employees, number of underground workers, total number of loaders, number of machine men, number and type of machines, depth of undercutting, number of shots per shift, number of shot firers, total daily wages of shot firers, size of cartridge used, method of firing, and average tonnage produced per 25-pound box of permissible explosive.

^a A joint State powder commission is provided for in the agreement between the Illinois coal operators and the miners for the settlement of any differences that can not be settled locally when a change of powder is to be made. The work of this commission includes settlement of disagreements relating to both black blasting powder and permissible explosives. Powder questions are first taken up locally by a local committee provided for under the same agreement and consisting of six representatives; three for the miners and three for the operators. If any disagreement can not be settled locally, it then goes before the joint powder commission for final decision. See pages 95-97 for the duties of this commission, agreement relative to settlement of powder questions, and procedure in making powder tests.

^b Quarterly Bulletin, The Illinois Coal Operators Association, vol. 6, 1914, p. 113.

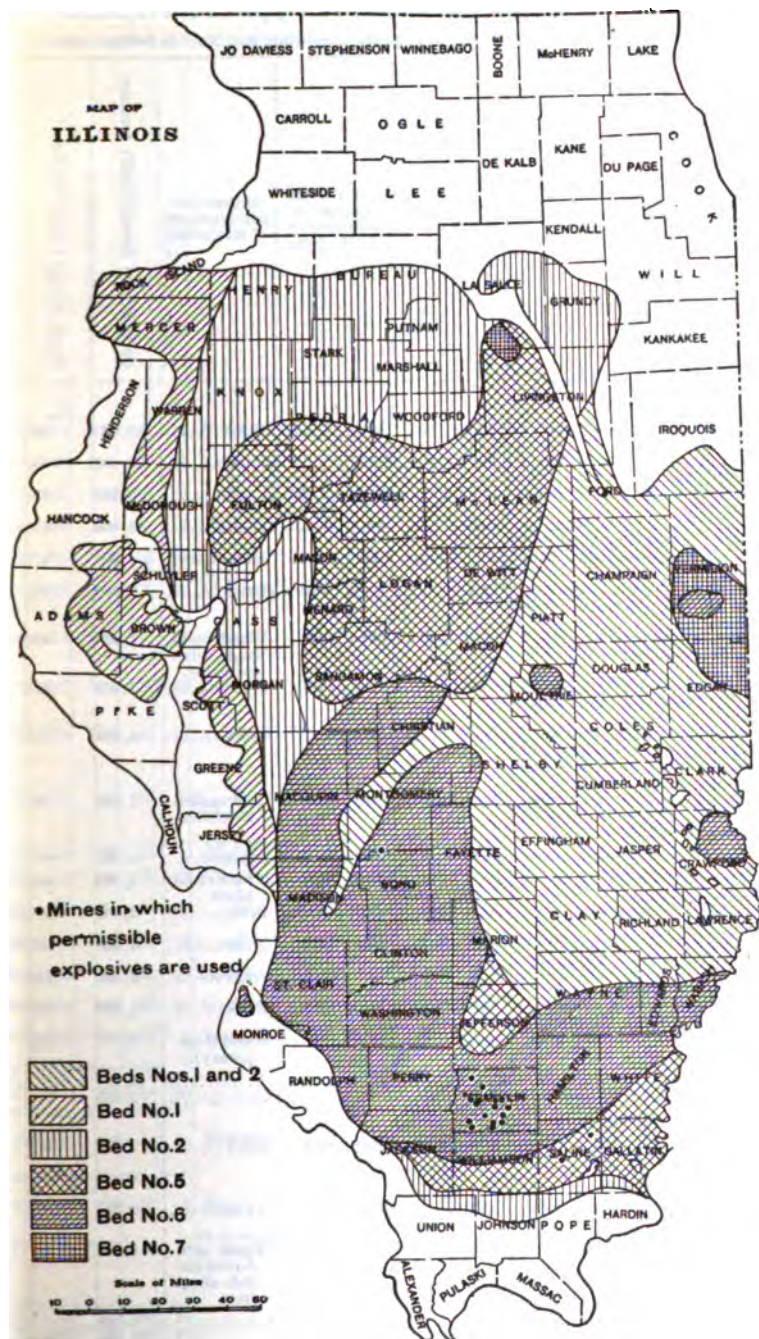


FIGURE 2.—Map of Illinois coal field, showing location of mines in which permissible explosives are used.

TABLE 1.—*Companies using permissible explosives exclusively*
 [Coal No. 6 mined in Franklin and Montgomery Counties, coal No. 5 in Saline County.]

Operator, mine, and nearest town.	Date of opening of mine.	Date of beginning use of permissible explosives.	Material of construction of magazine.	Approximate average, daily tonnage.	Depth of coal $\pm$ below surface.	Thickness of coal $\pm$ a bed.
<i>Franklin County.</i>						
United Coal Mining Co., No. 1 mine, Christopher.	Sept., 1905..	1909.....	Brick.....	3,000	<i>Feet.</i> 500	<i>Ft. in.</i> *9 6
Hart-Williams Coal Co., ^b No. 1 mine, Benton.	Sept., 1907..	1909.....	do.....	2,400	630	9 6
Benton Coal Co., ^b No. 1 mine, Benton.	1905.....	1909.....	do.....	2,500	634	9 6
Producers Coal Co., No. 18 mine (formerly Dering), West Frankfort.	1907.....	1910.....	do.....	1,300	510	10
United Coal Mining Co., No. 2 mine, Buckner.	Aug., 1910..	Aug., 1910..	Hollow tile..	4,000	472	*9 6
Producers Coal Co., No. 19 (formerly Dering No. 11) mine, West Frankfort.	1904.....	Sept., 1910..	Brick.....	2,000	494	10
Old Ben Coal Corporation, No. 8 mine, West Frankfort.	Sept. 1910...	Sept., 1910..	Wood, covered with tar paper.	4,400	460	*9 6
Bell & Zoller Mining Co., No. 1 (formerly Zeigler Coal Co.) mine, Zeigler.	1903.....	Dec., 1910..	Cement.....	3,800	417	12
Christopher Coal Mining Co., No. 1 (Old North) mine, Christopher, formerly operated by Zeigler District Collieries Co.	1906.....	1911.....	Brick.....	3,000	518	9 6
John A. Logan Coal Co., No. 1 mine, Logan (formerly Benton District Coal Co.), Hanaford.	Jan., 1909...	July, 1911...	Corrugated sheet-iron.	1,000	700	7
Sesser Coal Co., Sesser mine.....	1906.....	Dec., 1911...	Concrete....	3,000	647	8 6
West Frankfort Coal Co., No. 1 mine, West Frankfort.	Sept., 1912..	Aug., 1913..	Concrete block.	1,700	443	9 6
W. P. Rend Collieries Co., No. 1 mine, Rend.	Jan., 1906...	Sept., 1913..	Brick.....	2,000	567	7 8
Christopher Coal Mining Co., No. 2 (New North) mine, Christopher.	Mar., 1914...	Mar., 1914...	do.....	3,500	593	*10
Old Ben Coal Corporation, No. 9 mine, West Frankfort.	Oct., 1914...	Oct., 1914...	Concrete....	3,800	468	10
Chicago, Wilmington & Franklin Coal Co., No. 1 mine, Orient.	Dec., 1914...	Dec., 1914...	Brick.....	3,800	550	10
Middle Fork Mining Co., New mine, Benton.	1915.....	1915.....	Wood (temporary).	(^d)	600	*8
Total.....				45,200		
Average.....				2,825	541	9 5
<i>Montgomery County.</i>						
Shoal Creek Coal Co., No. 1 mine, Panama.	Sept., 1905..	May, 1913...	Brick.....	2,500	381	7
<i>Saline County.</i>						
Saline County Coal Co., No. 3 mine, Harrisburg.	1911.....	July, 1914...	do.....	2,700	270	7
Eldorado Coal & Mining Co., Seagraves mine, Eldorado.	1908.....	Apr., 1916...	Wood and corrugated sheet iron.	2,000	450	4 10
Total.....				4,700		
Average.....				2,350	360	5 11
Grand total.....				52,400		
Grand average.....				2,750	515	8 11

^a From annual coal report of Illinois, 1915, except those marked with an asterisk (*).

^b Some black powder used in drawing pillars in 1911.

^c Double shift.

and pertinent data as to methods. Data to Apr. 1, 1916.

[Coal No. 6 mined in Franklin and Montgomery Counties, coal No. 5 in Saline County.]

Number of employees. ^a		Total number of loaders. ^a	Machine men and helpers. ^a	Number of machines.	Type of machine.	Depth of undercutting.	Approximate number of shots per shift.	Number of shot firers.	Total daily wages of shot firers.	Size of cartridge used.	Method of firing.	Average tons produced per 25-pound box of permissible explosive per year, including rooms and entries.
Total.	Underground.											
451	412	230	38	19	Breast.....	6	250	4	18.84	1½ by 7 inches.	Fuse and No. 6 detonator.	140
568	518	305	36	18	do.....	7	200	4	18.84	1½ by 6 inches.	do.....	130
570	510	325	38	19	do.....	6	260	4	18.84	do.....	do.....	145
252	228	150	30	15	do.....	6	120	2	9.42	do.....	do.....	126
620	576	342	52	26	do.....	6	320	4	18.84	1½ by 7 inches.	do.....	130
300	275	146	66	33	Air puncher.	4	165	3	14.13	1½ by 6 inches.	do.....	125
548	471	350	54	27	Breast.....	6	320	6	28.26	1½ by 7 inches.	Battery and No. 6 electric detonator.	164
680	617	394	57	29	do.....	6	300	6	28.26	1½ by 6 inches.	Fuse and No. 6 detonator.	144
470	419	247	46	23	do.....	6	260	4	18.84	1½ by 6 inches, 1½ by 7 inches.	do.....	170
167	141	100	20	10	do.....	5	110	2	9.42	1½ by 6 inches.	do.....	128
483	463	325	32	20	Air puncher, shortwall.	5.5	250	4	18.84	do.....	do.....	202
287	262	185	26	13	Breast.....	6	130	2	9.42	do.....	do.....	138
434	394	256	41	21	do.....	5	240	4	18.84	do.....	do.....	120
450	408	289	38	19	do.....	7	250	4	18.84	do.....	do.....	176
445	400	300	32	16	do.....	7	240	4	18.84	1½ by 7 inches.	Battery and No. 6 electric detonator.	150
488	428	300	30	15	Shortwall...	7	225	4	18.84	1½ by 6 inches.	Fuse and No. 6 detonator.	140
*110	*95	*66	*12	6	Breast.....	6	Just developing.				do.....	
7,319	6,628	4,360	648	336			3,640	61	287.31			
430	390	256	38	20		6.3	228	4	18.00			145
504	546	335	44	22	Breast.....	6	300	4	18.84	1½ by 7 inches.	Fuse and No. 6 detonator.	107
425	384	224	36	19	Breast.....	6	225	4	20.06	1½ by 6 inches.	do.....	215
311	298	*205	24	12	(4 shortwall, 8 breast)	6.5	220	4	19.76	do.....	do.....	(e)
726	682	429	60	31			445	8	39.82			
368	341	215	30	11		6	223	4	19.91			215
8,649	7,856	5,124	752	379			4,385	73	345.97			
422	362	256	38	19		6+	231	4	18.21			147

^a Producing 600 tons daily; just being developed.^e First month's production, 250 to 290 tons in rooms, 150 tons in entries.

In the mines listed in Table 1, permissible explosives are now used exclusively. As noted in the table, the total average daily tonnage of the 19 producing mines is 52,400 tons, or 2,750 tons per mine. The total number of employees is 8,649, or more than 11.44 per cent of the total number, 75,607, employed in the mines of the State.^a

Of the mines listed, those in which permissible explosives have been used exclusively from the start are: Mine No. 2, United Coal Mining Co., Buckner; Mine No. 8, Old Ben Mining Corporation, West Frankfort; Mine No. 9, Old Ben Mining Corporation, West Frankfort; Mine No. 2, Christopher Coal Mining Co., Christopher; Mine No. 1, Chicago, Wilmington & Franklin Coal Mining Co., Orient; New mine, Middle Fork Mining Co., Benton.

Tests have been made at the No. 6 mine of the Saline County Coal Co., Grayson, in the No. 5 seam, and permissible explosives were used in all entries on trial during the month of March, 1915. Good results were obtained and the tests showed that permissible explosives would be cheaper for the miner. The company is anxious to introduce permissible explosives at this mine on account of the numerous fires resulting from the use of black powder. Since the company took over this property in 1914, solid shooting has been supplanted by machine mining. Although past experience has shown the danger of using black blasting powder, the acceptance of permissible explosives by the miners has been delayed, and the matter has been referred to the joint State powder commission.

In addition to the mines listed, permissible explosives have been used to some extent in other mines in Illinois for special work. Among these, the Wasson Coal Co. used some in its new slope mine No. 2, at Carriers Mills, in 1914, in development work when the coal was first reached. In 1915, the Canton Coal Co., Canton, used a small amount of these explosives in rock work; the Marion County Coal Co., Centralia, used about 200 pounds in a test in blasting coal, and the New Enterprise Coal Co., used 1,725 pounds for shooting wet holes at their stripping operation at Marion, Ill. Permissible explosives have been used in some entry work at No. 1 mine, Royalton, but have not come into general use throughout the mine. Of the 20 mines in Franklin County, all use permissible explosives exclusively, except two at Royalton, and one at Freeman.

COST OF PERMISSIBLE EXPLOSIVES TO OPERATOR AND TO MINER.

It has been the custom for many years in Illinois mines for the operators to sell the black blasting powder to the miners, at a price that has been specified in the jointly made annual or biennial contracts between operators and miners.

^a Annual coal report of Illinois for 1915, State Mining Board, 1916, p. 2.

The selling price of permissible explosives during the introductory period ranged from \$3 to \$3.50 per 25-pound box. In some instances, so it was claimed, this increased the cost to the miner about 1 cent per ton of coal, whereas in other instances there was no marked difference compared with black blasting powder selling at \$1.75 per 25-pound keg, because the production with permissible explosives was found to be approximately in the proportion of 2 tons to 1 with black blasting powder. As soon as the success of permissible explosives was assured the miners raised the issue as to their rights in the establishment of the price of the explosive to come under the operation of their contract. In making his report to a State-wide meeting of the miners Groce Lawrence made special mention of the introduction of permissible explosives during the fall and winter of 1909 and 1910, as follows:^a

The United Coal Mining Co. at Christopher was experimenting with a permissible explosive. At Sesser, the Franklin County Collieries Co. was making a test of another permissible explosive, and at Rend City the Rend Coal Co. was experimenting with some other brand.

Board Member Smith and myself had this permissible explosive question up with these operators on several different occasions prior to April 1, 1910, but nothing definite was done other than to inform them and the miners in their employ that if they wished to change from the powder then in use to this new explosive they would have to file a petition and handle the proposition in its entirety as provided for in the eighth clause of the State agreement.^b

On several occasions the miners of Franklin County spoke to me about buying and using these new explosives. I always took the time and pains to explain to them what their rights were under the contract, and advised them not to buy any of it until the question had been definitely settled and a price agreed upon. * * *

Immediately after the settlement was made last September a petition was filed by the miners working at the No. 1 mine of the United Coal Mining Co. at Christopher to have the [joint State] powder commission make a test at that mine to determine what the relative cost of permissible explosive would be as compared to black powder at \$1.75 per [25-pound] keg.

As soon thereafter as arrangements could be made, or on September 28, 1910, the joint powder commission met at Christopher for the purpose of making a test at the No. 1 mine of the United Coal Mining Co. They devoted about 30 days to the making of this test and in trying to reach an agreement as to the relative cost.

In this same report is related the difficulty attending the agreement as to the relative price, the miners contending for a price of \$2.45 per 25-pound box, whereas the operators, on account of the cost of permissible explosives and certain other facts as to relative productive values, endeavored to obtain a price more satisfactory to them.

^a Twenty-second Annual Convention of District No. 12, United Mine Workers of America, 1911, pp. 33, 34.

^b See p. 96.

A temporary agreement was made as follows:

TEMPORARY AGREEMENT OF JOINT POWDER COMMISSION, OCTOBER 12, 1910.

Meeting of the joint powder commission,^a St. Louis, Mo., for the purpose of taking up and considering the data and all matters pertaining to the test made with (name of permissible explosive) and black blasting powder at the United Coal Mining Co.'s No. 1 mine, Christopher, Ill. The commission agree to use permissible explosive in the above-mentioned mine, and based on a temporary price of \$2.80 a box of 25 pounds of explosive to be charged the miners until a more rigid and thorough investigation and test be made, and which must be made within 30 days from date, and should any charge be made other than \$2.80 per box after a permanent price has been settled upon by the commission that operator and miners shall be paid said difference as the case may be.

The commission was agreed that the use of permissible explosives would be for the benefit and protection of life and property. The matter was finally adjusted as stated by C. M. Moderwell,^b president of the Illinois Coal Operators' Association:

The temporary agreement of the joint State powder commission made October 12, 1910, fixed a price of \$2.80 per box on the permissible explosive which was then being used at the United Coal Mining Co.'s No. 1 mine at Christopher. It was decided by the mining company that the use of a permissible explosive was absolutely necessary at this mine, but the attempt to introduce it caused a good deal of trouble, and while the temporary agreement above referred to was in effect for some time, on January 17, 1911, the joint powder commission reported a final disagreement and referred it to the joint executive board of the miners and operators. Early in March, 1911, the miners' officials issued orders to the miners working in the mine above referred to, as well as those in other mines in that district, not to pay more than \$2.45 per 25-pound box of carbonite or other permissible explosive after April 1. A special committee representing Franklin County met the miners' board in Chicago on April 7, 1911, and D. W. Buchanan, for the operators, and John H. Walker, for the miners, were appointed to investigate the prices charged for permissible explosives in the East. They reported at a meeting of the joint executive board held in Chicago May 10 and 11, and the following was finally agreed upon:

In case any operator desires to use a permissible explosive, a price of \$2.45 per box for (names of two brands of permissible explosives) is hereby established during the remainder of the present contract year, it being understood that the efficiency of the "permissible" used is to be maintained equal to the same standard established in the joint test upon which the above price was based. Any dispute arising out of this decision as to the efficiency of the "permissible" furnished shall be taken up the same as any other powder dispute under the contract.

The price of \$2.45 per 25-pound box has since been made a part of the contract and is operative at the present time. Although this allows a small margin of profit compared with handling black blasting powder, there has been no dissatisfaction. However, in 1915 and 1916, owing to the European war, manufacturers' prices of permissible explosives were raised, so that some coal companies were selling permissible explosives at a loss of from 50 to 75 cents per 25-pound box. Some discussion was given the question of

^a See pp. 96-97 for duties of commission.

^b By correspondence.

equalizing the losses due to the increased cost of explosives during the joint convention in the establishment of the new wage scale, April, 1916, but no decision was reached, and the original contract is in force. Explosive is delivered to the miner at his working place at \$2.45 per 25-pound box.

KINDS OF PERMISSIBLE EXPLOSIVES USED IN ILLINOIS.

There are at present four powder companies selling permissible explosives in Illinois—the Aetna Explosives Co. (Inc.), New York, N. Y.; the Burton Powder Co., Pittsburgh, Pa.; E. I. du Pont de Nemours & Co., Wilmington, Del.; and the Illinois Powder Manufacturing Co., St. Louis, Mo. The explosives and their characteristics are given in Table 2 following. As noted, the nitroglycerin class of explosives is used almost exclusively in Illinois.

TABLE 2.—Permissible explosives used in Illinois and their characteristics.^a

Brand and manufacturer.	Size of cartridge, inches.	Rate of detonation, ^b feet per second in 1½-inch cartridge.	Pressure, ^c pounds per square inch.	U. D. C., ^d grams.	Compression, small lead block test, ^e millimeters. <i>f</i>	Expansion, traulz lead block test, ^g cubic inches.
Aetna Explosives Co. (Inc.):						
Aetna coal powder B.....	1½ by 6....	9,870	^h 84,083	301	13.8	11.90
Aetna coal powder C.....	1½ by 6....	7,010	60,140	344	10.7	8.78
Burton Powder Co.:						
Mine-ite B.....	1½ by 7....	9,540	62,080	340	10.5	9.70
E. I. du Pont de Nemours & Co.:						
Carbonite No. 3.....	1½ by 7....	8,710	^h 70,742	335	10.8	9.52
Carbonite No. 5.....	1½ by 7....	10,140	70,270	304	11.7	9.52
Carbonite No. 6.....	1½ by 7....	7,490	62,470	345	8.8	7.38
Monobel No. 5 ⁱ	1½ by 6....	6,790	106,100	267	12.2	8.97
Illinois Powder Manufacturing Co.:						
Black Diamond No. 3-A.....	1½ by 6....	11,160	82,590	304	14.5	10.13
Black Diamond No. 6-L F.....	1½ by 6....	9,630	88,770	306	12.2	9.76
Average.....		8,920	76,355	314	11.8	9.52

^a See Bulletin 66, Tests of permissible explosives, by Clarence Hall and Spencer P. Howell, for significance of terms and detailed results of tests.

^b Rate of explosion, or rate at which detonation travels through a given length of explosive. Rate of detonation of 1½-inch diameter cartridges greater, but not yet determined by experiment.

^c Maximum theoretical pressure exerted on explosion, in paper wrapper except where noted.

^d Unit deflective charge. Number of grams required to give same propulsive effect as 227 grams of 40 per cent standard dynamite.

^e A certain measure of relative compressive effect of explosive, with charge unconfined.

^f 25.4 millimeters—1 inch.

^g A certain measure of comparative disruptive effect of explosive.

^h In tinfoil wrapper.

ⁱ All explosives are Class 4, nitroglycerin, excepting Monobel No. 5, which is Class 1a, ammonium nitrate, containing nitroglycerin.

NOTE.—The weight per stick of explosive varies, there being 40 to 53 sticks per 25-pound box of explosive.

GENERAL INCREASE IN THE USE OF PERMISSIBLE EXPLOSIVES IN ILLINOIS AND THROUGHOUT THE UNITED STATES.

The following figures from the annual coal reports of Illinois show the marked increase in the quantity of permissible explosives used in coal mining in Illinois during the fiscal years ended June 30, 1911, to 1915: 1911, 243,099 pounds; 1912, 328,075 pounds; 1913, 603,420

during the winter months for thawing and storing each day's supply of explosives. The thaw houses are heated by exhaust steam at low pressure. Usually, however, the mine temperature, which normally in the interior of Illinois coal mines is 60° to 70° F, is depended upon to thaw the explosive and keep it at a proper temperature for use.

The Bureau of Mines has made a special study of the storage and handling of explosives, and attention is called to the bureau's Technical Paper 18,^a which deals with the problems of selection of site and specifications of construction of magazine. The type recommended by the bureau is shown in Plate IV, A, and details of construction in figure 3. Such information may serve as a guide to future construction of magazines at Illinois mines.

The following brief comment is taken from Technical Paper 18:^b

BUREAU OF MINES MAGAZINE.

As a result of experiments and of information furnished by the manufacturers of explosives, a cement-mortar magazine (Pl. IV, A, and fig. 3) has been erected by the Bureau of Mines. The magazine has a capacity of 20,000 to 30,000 pounds of explosives, and was built at a cost of \$400. The outside dimensions are 10 to 14 feet.

The salient features of the magazine are the cement-mortar walls, sliding door, and roof. The cement mortar is 6 inches thick in all walls and 3 inches thick in the roof and the door. The door is secured by two substantial locks. No metal of any kind is exposed on the inside of the magazine. The ventilators above the floor are arranged to prevent the entrance of bullets or firebrands.

The means provided for ventilation have been found to be adequate, and, accordingly, the storage conditions are favorable for keeping the explosives from deteriorating. The cement-mortar construction is effective in resisting the penetration of rifle bullets, and because of its friable nature offers an additional advantage for the reason that, in the event of an explosion in or near the magazine, large masses of material will not be projected over the surrounding country. The galvanized-iron covering is fire resisting; it also serves as an excellent medium for protection against lightning, as the four corners of the building are properly grounded with metal rods.

In order that the details of construction may be thoroughly understood, the following bill of materials is presented. The concrete of the foundation consists of one part cement, three parts sand, and five parts gravel. The cement-mortar in the walls, door, and roof consists of one part cement to six parts coarse sand

BILL OF MATERIALS.

Lumber:

- 1 piece of yellow pine, 2 inches by 6 inches by 18 feet.
- 1 piece of yellow pine, 6 inches by 8 inches by 14 feet.
- 11 pieces of yellow pine, 2 inches by 10 inches by 10 feet.
- 2 pieces of yellow pine, 1 inch by 3 inches by 12 feet.
- 4 pieces of yellow pine, 2 inches by 10 inches by 10 feet.
- 8 pieces of yellow pine, 2 inches by 6 inches by 14 feet.
- 9 pieces of yellow pine, 2 inches by 8 inches by 10 feet.
- 6 pieces of yellow pine, 2 inches by 4 inches by 7 feet.
- 6 pieces of yellow pine, 2 inches by 4 inches by 8 feet.
- 2 pieces of yellow pine, 4 inches by 4 inches by 7 feet.

^a Hall, Clarence, and Howell, S. P., *Magazines and thaw houses for explosives: Technical Paper 18, Bureau of Mines, 1912, 34 pp.*

^b Hall, Clarence, and Howell, S. P., work quoted pp. 18, 19, 25, 26.



A. WOODEN POWDER MAGAZINES TYPICAL OF THOSE IN SOUTHERN ILLINOIS.



C. CONCRETE-BLOCK POWDER MAGAZINE AT AN ILLINOIS MINE.



B. POURED-CONCRETE MAGAZINE AT AN ILLINOIS MINE.



D. BRICK POWDER MAGAZINE AT AN ILLINOIS MINE.



A. BUREAU OF MINES CEMENT-MORTAR MAGAZINE.



B. BUREAU OF MINES CEMENT-MORTAR THAW HOUSE.

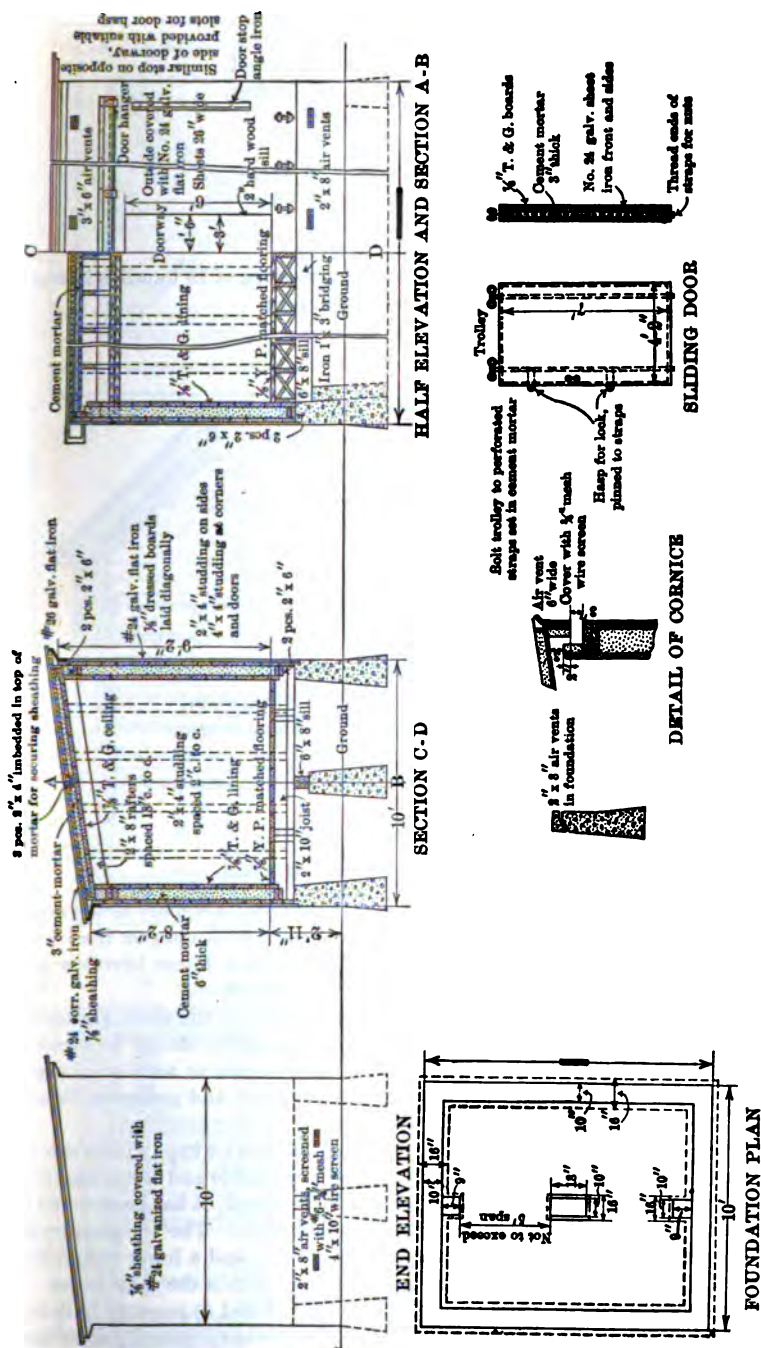


FIGURE 3.—Plan and sections of Bureau of Mines cement-mortar magazine.

Lumber—Continued.

- 2 pieces of yellow pine, 4 inches by 4 inches by 8 feet.
- 24 pieces of yellow pine, 2 inches by 4 inches by 9 feet.
- 3 pieces of yellow pine, 2 inches by 4 inches by 14 feet.
- 9 pieces of hemlock, 1 inch by 10 inches by 10 feet.
- 8 pieces of hemlock, 1 inch by 10 inches by 12 feet.
- 15 pieces of hemlock, 1 inch by 10 inches by 14 feet.
- 28 pieces of hemlock, 1 inch by 10 inches by 16 feet.
- 400 feet of lumber for framework of foundation.
- 48 board feet of 16-foot No. 2 white-pine flooring.
- 856 board feet of No. 2 yellow-pine flooring, consisting of 19 bundles 16 feet long, 1 bundle 14 feet long, and 11 bundles 10 feet long.

Hardware:

- 100 pounds of 8d. wire nails.
- 40 pounds of 20d. wire nails.
- 4 pieces of iron, $\frac{3}{4}$ inch by $1\frac{1}{2}$ inches by 7 feet.
- 2 pieces of iron, $\frac{3}{4}$ inch by $1\frac{1}{2}$ inches by 18 inches.
- 2 pieces of angle iron, 3 inches by 5 feet long.
- 2 pairs of No. 130 Coburn trolley hangers.
- 8 feet of No. 4 track.
- 2 end brackets.
- 1 center bracket.
- 1 heavy iron door handle.
- 4 brackets with hardwood rollers.
- 500 square feet of No. 26 gage galvanized flat iron.
- 150 square feet of galvanized corrugated iron.
- 2 pieces of 5-inch by 8-inch, $\frac{1}{2}$ -inch mesh, No. 6 wire screen.
- 8 pieces of 4-inch by 10-inch, $\frac{1}{2}$ -inch mesh, No. 6 wire screen.

Cement, sand, and gravel:

- 80 bags of cement.
- 200 bushels of sand.
- 150 bushels of gravel.

CONSTRUCTION OF THAW HOUSES.

What has been said of the construction of magazines applies equally to the designing of thaw houses. In addition to protecting thaw houses from dangers from without, the increased sensitiveness of explosives within the thaw house involves another danger, due to the high temperatures that must be maintained.

Thaw houses are usually situated so as to be convenient to the shaft, pit mouth, or place where the explosives are being used, and consequently should be constructed of material that would not be projected in large pieces should an accidental explosion occur. It is also essential that thaw houses be bullet proof, and protected from lighting, unlawful entry, and fire.

In constructing thaw houses some source of heat that can be kept within safe limits must be introduced. Low-pressure steam is usually available and when used at pressures not exceeding 3 pounds, in a manner herein described, it has been found to be effective and one of the safest means of thawing explosives. The temperature of the air entering a compartment should never exceed 130° F., and a lower temperature is desirable if a temperature of 90° F. can be maintained within the thaw house.

The cement-mortar thaw house (Pl. IV, B, and figs. 4 and 5) recently built by the Bureau of Mines for thawing large quantities of explosives has a capacity of 500 pounds and cost complete \$200. It was erected after a consideration of the various details involved, is not of prohibitive cost, and offers the following advantages:

The cement-mortar walls, roof, and doors protect the explosives from bullets; the cement-mortar and galvanized-iron covering are fire resisting, and the latter furnishes

a good conductor for lightning; the building is substantial; entrance is difficult and is unnecessary, for the trays can be entirely removed without anyone entering the house (see Pl. IV, B); all parts of the house are accessible for cleaning; the explosives can be distributed in thin layers in the trays; the thermometers are easily read from without.

The thaw house of the Bureau of Mines also offers the following advantages in regard to temperature: No high temperature from artificial sources of heat obtains in the vicinity of the thaw house; the low-pressure steam, hot-water, or electric coils are not in the thaw house proper but in a separate compartment at its side, and hence particles of explosives can not come in contact with them; the explosives are not subjected to the evil effects of free steam or water; the temperature is surely and easily controlled with little attention; the entrance of cool air or the escape of heat from the thaw house is reduced to a minimum; the stack makes possible the positive circulation of heated air.

DELIVERY AND STORAGE OF EXPLOSIVES UNDERGROUND.

DELIVERY TO MINER.

The miner usually orders his explosives at the mine office, signing a slip, a copy of which is pasted on the box of explosive to be delivered. The inscription on one of these slips is given below:

Order slip for permissible explosives.

Side.....	Mine No.....
Entry No.....	Deliver to..... Ck. No.....
Room.....	One Box Permissible Explosive
Date.....	Name.....

The explosive for the following day is taken from the powder magazine after quitting time. After the shots have been fired, the supply ordered is taken below and distributed to various room necks and entries by a powder man with mule and car while the electric current is turned off. Explosive is delivered in 25-pound boxes. The State law ^a reads as follows:

Amount of powder kept in mine.—(a) No blasting powder or other explosives shall be stored in any coal mine, and no workmen shall have at any time in the mine more than 35 pounds of black powder nor more than 25 pounds of permissible explosives, nor more than 3 pounds of other high explosives, provided that nothing in this section shall be construed to prevent the operator of any mine from taking into the mine, when miners are not therein, and in electrically equipped mines, while the current is turned off on roadways through which it is transported, a sufficient quantity of powder for the reasonable requirements of such mine for the next succeeding working day. The delivery of powder into coal mines shall be during the interval after the shot firers have come out of the mine and prior to the entry of the day shift into the mine in the morning; but in the interim before such powder is delivered to the men, it shall be kept in a closed receptacle.

Explosives shall not be carried in the same car with tools or other materials.

As the working agreement ^b places the responsibility of delivery of explosives on the operator, some mines have a system of storing explosives at various partings or sections in the mine in wooden

^a Coal-mining laws of Illinois, sec. 19.

^b See "working agreement between operators and miners," p. 97.

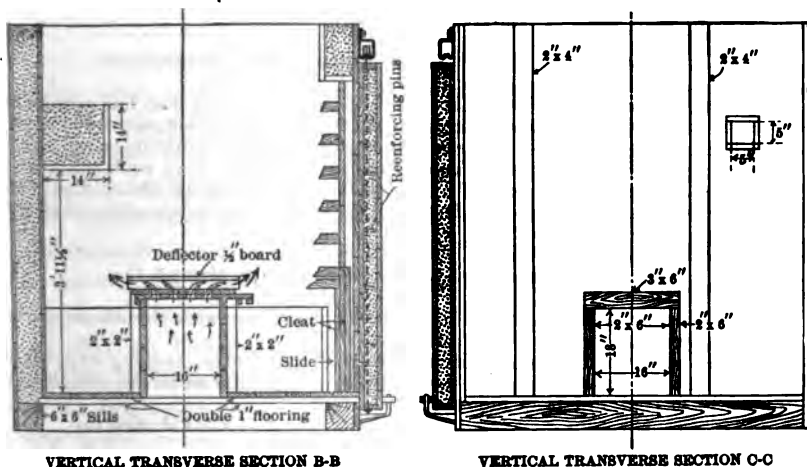
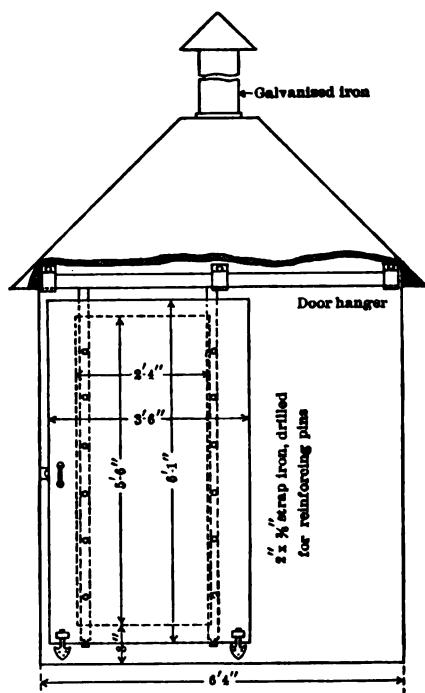
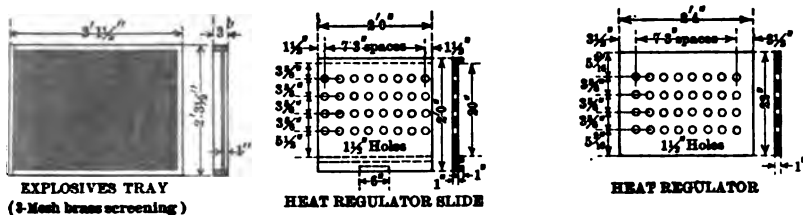


FIGURE 5.—Elevation and details of Bureau of Mines cement-mortar thaw house.

chests with a capacity of about 220 pounds, the explosive being taken in and stored after the shift has left the mine. Explosive is then delivered to miners at their working places by the powder man on the following morning and a receipt obtained. Another method of designating the owner is to paint the miner's check number in red on the end of the box. The amount of explosive delivered per day varies, usually the greatest amount being at the beginning of a two-weeks' pay, as the men will often postpone ordering for a day or two when the close of a two-weeks' period is near. In some of the larger mines the total amount of permissible explosive used per day averages 700 pounds.

USE OF POWDER BOXES.

The Illinois coal-mining law states that the explosive must be kept in a wooden box, securely locked, with a hinged lid. The boxes are usually kept in the first or second crosscut from the face. The law does not specify the dimensions of the powder box to be used, but for the most part boxes made of 1-inch board, of convenient shape and large enough to hold the allowable amount of explosive, are used. A great diversity of boxes exists; probably about three-fourths of the boxes in use fully comply with the requirements of the mining law. Some mines show marked superiority over others in this respect, and, to a certain extent, the conditions of storage of explosives reflect the attitude of officials in the enforcement of safety measures.

Mine inspectors and officials are constantly urging compliance with the law in detail, and particularly that boxes should never be placed where power cables may be laid in proximity to them. The following notice is an example of the efforts being made in this direction:

NOTICE TO EMPLOYEES.

EXTRACT FROM ILLINOIS STATE MINING LAWS (SECTION 19).

"Place and manner of keeping in the mine.—(b) Every person who has powder or other explosives in a mine shall keep the same in a wooden box, securely locked, with hinged lid, and said box shall be kept as far as possible from the track; and all powder boxes shall be kept as far as practicable from each other and each in a scheduled place. Black powder and high explosives or caps shall not be kept in the same box. Detonating explosives and detonators shall not be kept in the same box."

NOTICE TO EMPLOYEES OF THE ——— COAL MINING CO.

This company will require an immediate compliance with the above provisions of law.

Miners must provide themselves at once with lock boxes, keeping explosives and caps in separate boxes, and keep boxes in crosscuts as far as practicable from face of coal and from motor tracks.

———— Coal Mining Co.,
 ————— Gen. Mgr.

Tack this notice on your powder box.

DELIVERY AND STORAGE OF FUSE AND DETONATORS.

Necessary fuse and detonators are carried into the mines by the individual miners. Fuse is taken into the mine in rolls of 50 to 100 feet. Double tape fuse is used almost exclusively. This is an advantage, for even in dry mines wet holes are occasionally encountered, and a cheaper grade of fuse would be false economy. During the slack summer season a roll of fuse may not be completely used in three to four months. The ingredients in fuse, both the powder and the hemp wrapping, readily absorb moisture, and if the fuse is exposed to the damp floor of the mine or the moist air current it will become damp and be liable to cause misfires or dangerous hangfires. Because of the safety and economy in proper storage, it would be desirable to supply an air-tight container, which could be furnished at small cost to each miner, suitable for holding a roll of fuse and a box of detonators.

DETONATORS.

Detonators, or blasting caps, are supplied at normal prices to the miners in boxes of 100 at a cost of approximately \$1 per box for the No. 6 grade. These are used almost exclusively in Illinois. The same quantity of No. 5 detonators can be purchased for \$0.15 to \$0.25 less per box than No. 6 at the local hardware stores, and miners sometimes buy them, but consistent results can not be obtained with No. 5 detonators. Powder men contend that an explosive detonates with greater strength when a No. 6 detonator is used, and strongly urge the exclusive use of detonators of not less strength than No. 6, which is one of the requirements of permissibility. With some permissible explosives a No. 7 detonator must be used. If the explosive has been stored for a period of approximately six months in the magazine, or if it has become damp in the miner's box underground or in a damp drill hole, the use of a No. 5 detonator is not only inefficient but dangerous. Under these conditions it is probable that a No. 5 detonator will not detonate the explosive properly, but will cause a misfire, or blow the stemming and part of the explosive out of the drill hole, or merely ignite the explosive, and start a fire or an explosion.

It is quite probable that detonators kept underground without careful storage for two or three months may become damp and weak. Tests have shown that when detonators become damp as little as one-sixth of the fulminate has been known to explode, which is insufficient to detonate a stick of explosive. The 100-detonator boxes are provided with a sheet of blotting paper which is supposed to keep out moisture, being laid on top of the caps and the cover placed over it. This sheet fulfills its purpose when the box is full, but after the detonators fall from the vertical position, it does not cover the ends of the detonators and furnishes little protection

from moisture. Some miners recognize this fact and stuff cotton or paper over the detonators in the box to remedy this defect; others who do not take such precautions find that they have frequent misfires. To offset this, R. M. Medill, superintendent of the Dering Coal Co., first introduced detonators in boxes of 25 each in Franklin County. Some companies have lately been furnishing detonators in moisture-proof packages of 10 each. The detonators are so packed that they do not come in contact. With this style of package the miner can always feel sure of having a strong detonator and the packages are safer to handle than an ordinary box of detonators. (See Pl. V, A.) They are furnished at a small increase in cost.

A box that is sometimes used contains a perforated partition which serves to hold the detonators upright in the box, even after most of them have been removed, and a piece of blotting paper covers their open end. A box of this type used in metal mines and quarries is illustrated in Bulletin 80^a of the bureau. Such a box can be used indefinitely, as the detonators can be sold loose to the miners in small quantities not to exceed 25 at a time.

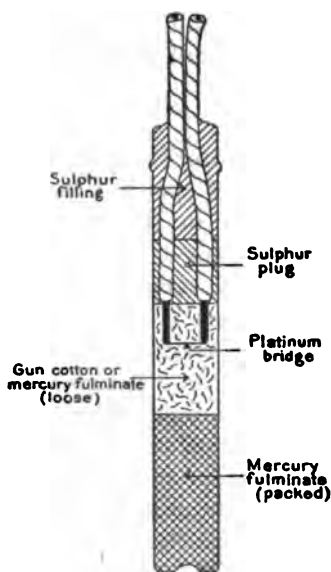


FIGURE 6.—Electric detonator, showing its component parts.

At a few mines the miners obtain their detonators at the shaft bottom. This is regarded as a much better practice than where the miners obtain them on the surface and then descend on the cage.

ELECTRIC DETONATORS.

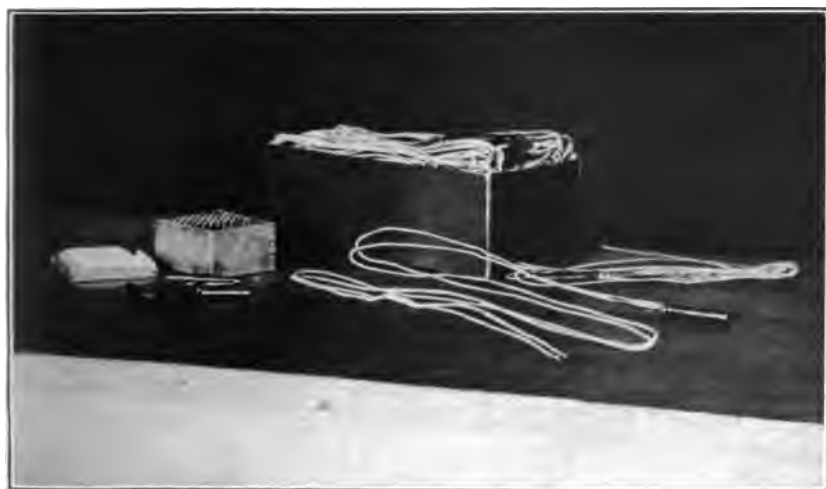
In two mines electric detonators are supplied to the miners in boxes containing 50 each. No. 6 electric detonators with 8-foot iron wires are used. No. 6 electric detonators with 8-foot copper wires were formerly used but have been practically replaced by the detonators with iron wires. (See fig. 6, which shows component parts of electric detonators.)

Many of the difficulties attending the use of fuse and detonators are eliminated by the use of electric detonators, as they are not easily affected by moisture and do not require any preparation by the miner. Plate V, B, shows boxes of detonators and electric detonators.

^a Munroe, C. E., and Hall, Clarence, A primer on explosives for metal miners and quarrymen; Bull. 80, Bureau of Mines, 1915, p. 41.



A. MOISTURE-PROOF PACKAGE HOLDING TEN DETONATORS.



B. DETONATORS AND ELECTRIC DETONATORS.



SYSTEMS OF BLASTING.

UNDERCUTTING AND "SNUBBING."

In the Illinois coal mines using permissible explosives the undercutting is done in the bottom coal adjacent to the under clay. In 18 of the mines listed all undercutting is done with breast chain machines. Only one of these mines uses the "short-wall" type of machine exclusively, which cuts across the face after first "sumping" in. Two other mines have a few "short-wall" machines. Puncher machines are used in two of the mines listed; in one of these all the coal is undercut entirely by puncher machines; in the other the output is about equally divided between puncher and chain machines.

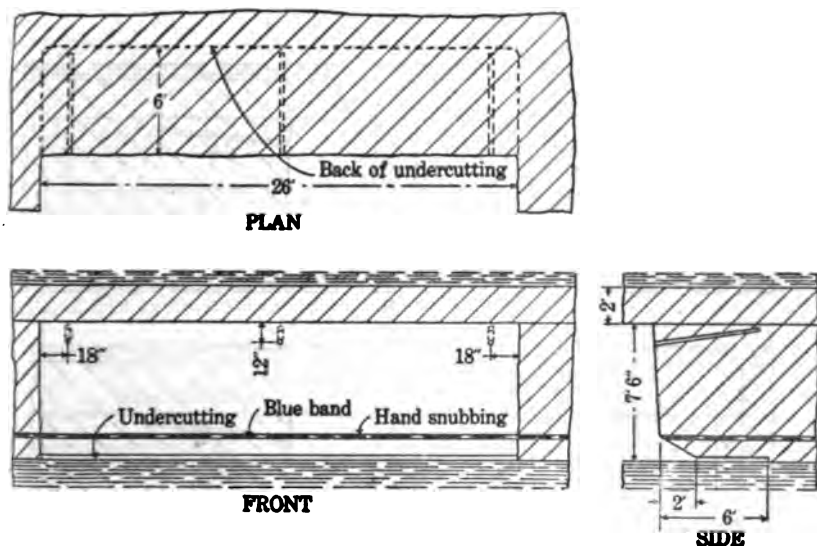


FIGURE 7.—Arrangement of drill holes, No. 6 seam, typical of practice with permissible explosives; coal undercut by chain machine, pick and wedge method of snubbing.

After the coal in the No. 6 seam has been undercut by chain machine it is generally "snubbed," either by sledge and wedge or by pick on a sloping face extending from the "blue band" to about one-half the depth of the undercut, as shown in figure 7. This system of snubbing previously followed in the Staunton field as early as 1897, was introduced in Franklin County by the agreement between Franklin County operators and miners. All cuttings are required to be loaded out before any shooting is done. Sometimes "snubbing shots" are fired.

Figure 8 shows methods of snubbing in entries. In entry work the most general practice is hand snubbing and shooting the coal with two holes, as shown at A. In one new mine where development

work has been progressing rapidly, the firing of one snubbing shot, as shown at B, has been used with good results. In the thicker coal two snubbing shots are fired in entries, as shown at C.

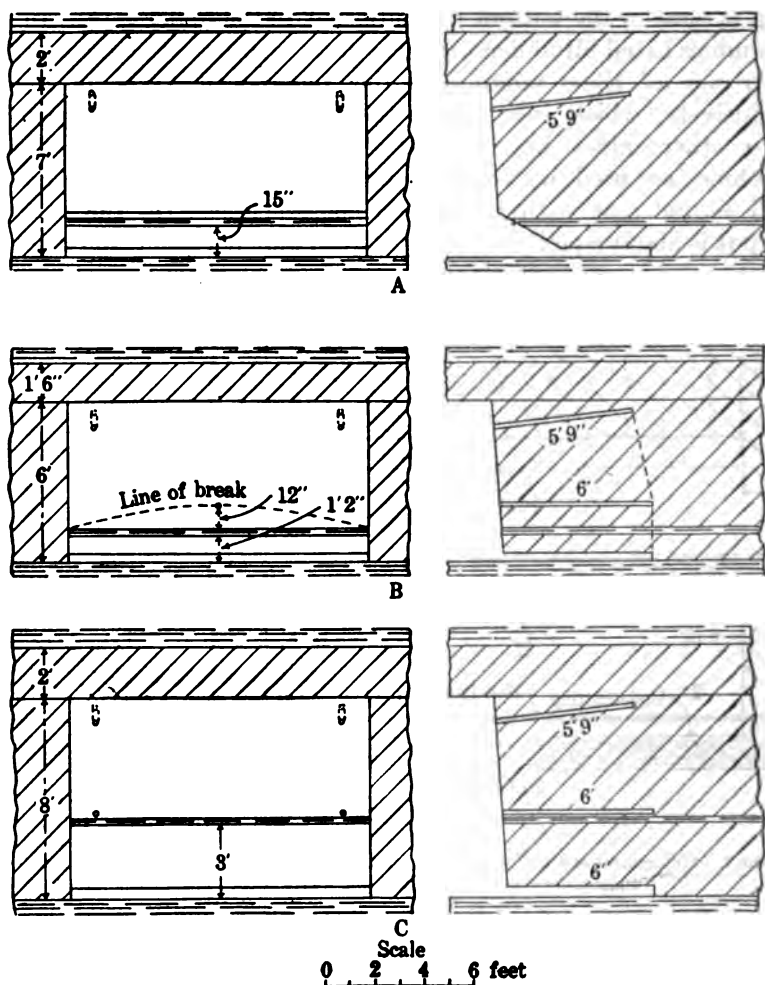


FIGURE 8.—Arrangement of drill holes in entry work, No. 6 seam, typical of practice with permissible explosives.

LOCATION AND ARRANGEMENT OF DRILL HOLES FOR PERMISSIBLE EXPLOSIVES.

The size of drill hole used in these mines averages $2\frac{1}{4}$ to $2\frac{1}{2}$ inches in diameter. The two common types of drills in use ("post" and "grip") are shown in Plate II, B and C.

There is an agreeable uniformity in the general arrangement of shots in the various mines in which permissible explosives are used. The working dimensions are uniform for the most part, rooms being

24 to 26 feet in width. The height of coal worked varies from 7 to 10 feet, $7\frac{1}{2}$ to 8 feet being the average; 10 to 40 inches of top coal is left for roof. A typical representation of the drill holes as to length, location, and arrangement in rooms for mines working the No. 6 seam in which the coal is undercut by chain machines is shown in figure 7; figure 8 gives similar details for entries. Figure 9 shows the arrangement of shots in mining the No. 5 coal, in which no top bench is left for roof.

In rooms the face is usually shot with three holes drilled parallel with the rib and about 12 inches from the roof, consisting of a center hole and two rib holes drilled about 18 inches from the ribs. The holes usually slope upward about 8° or 10° and are drilled to a depth equal to or slightly less than the depth of the undercutting, which varies

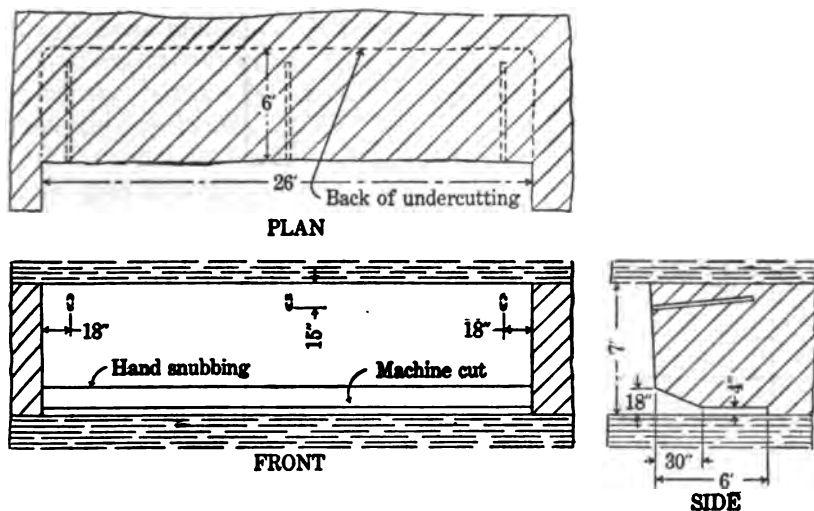


FIGURE 9.—Arrangement of drill holes, No. 5 seam, coal undercut by chain machine, hand snubbing.

from $4\frac{1}{2}$ to 5 feet with puncher machines and from 5 to 7 feet with chain machines.

There are a number of modifications of this method. In some instances holes are drilled as far as 2 to 3 feet from the rib and about 18 inches from the roof. In some mines horizontal holes are drilled, and these give good results, but usually on account of the height of the coal and the difficulty of drilling so close to the roof, the holes are given a slight angle. Undoubtedly the best results are obtained when the hole is only slightly inclined and the depth of the hole is about 6 to 12 inches less than the depth of undercutting. Usually in mining the coal breaks along certain faces independent of the angle of the drill hole, but in the No. 6 coal the cleat is not very pronounced. The advantage of inclining the holes is to cross the bedding planes which in some places are distinct. Drill holes should

not be deeper than within 6 inches of a vertical plane through the back of the undercutting. Holes drilled so that the point is close to the rib or is in solid coal at the back require more explosive and yield a smaller proportion of lump coal. In entry work in these mines generally two holes are used. Sometimes the hole for the "buster" shot, which is always given the shortest fuse in order to insure its being fired first, is drilled about 18 inches from one of the ribs, and the hole for the second shot is drilled 9 to 12 inches from the other rib. On the next round the position of the "buster" shot and the second shot is reversed, thus maintaining a square rib.

When black blasting powder was formerly used in these mines, five holes were fired as shown in figure 10. The two shots immediately

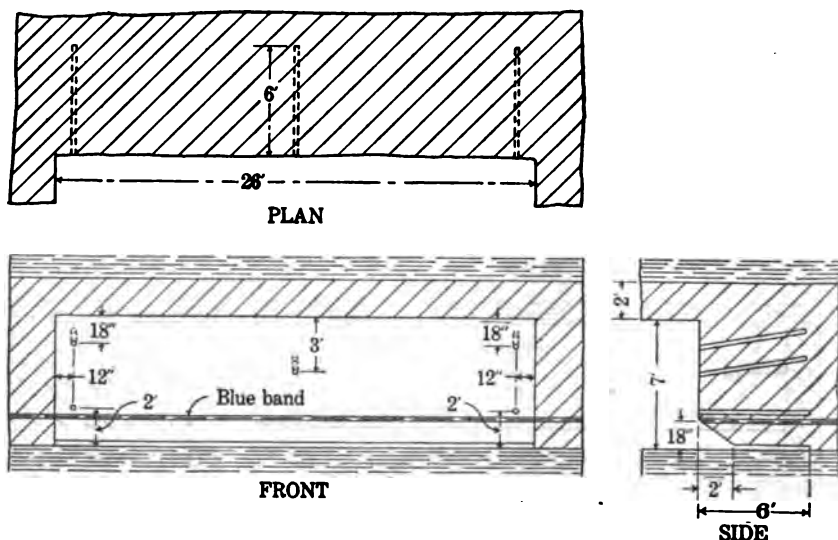


FIGURE 10.—Arrangement of drill holes, No. 6 seam, typical of practice in Franklin County when black blasting powder was used and snubbing was done by powder, in mines where permissible explosives are now used. Method of snubbing with permissible explosives used in coal 9 to 10 feet thick.

above the "blue band" acted as "snubbing" shots. At the outset a number of miners practiced the same method with permissible explosive, firing snubbing shots. This method is used to some extent at Zeigler and Sesser in mining thick coal—9 or more feet thick—and is strongly recommended as smaller charges of explosive can be used per shot and the limit charge will not be exceeded, but the general practice throughout the field is hand snubbing. In some places in the Zeigler-Christopher field the "blue band" becomes so thick, about 12 inches, that it is left in place, serving as the bottom. When the work to be done requires the use of an excessive amount of explosive in three holes, then a greater number of holes should be drilled.

Permissible explosives have been used in shooting off the solid to some extent in drawing pillars and have given good results. It has been found that these explosives can be used to good advantage in shooting out the bottom bench below the "blue band;" then the remaining coal is shot down as when undercut. Their use has worked satisfactorily in a number of instances in "slabbing" entries and in recovering top coal.

In a certain Franklin County mine, hand mining is done in some few scattered rooms which are too far removed from the machine districts to be economically handled on the machine basis. Usually expert miners are assigned to such places. Cutting is done by hand immediately above the middle bench in the brittle layers of coal about 2 feet from the coal roof. Cutting is readily accomplished to a depth of 5 to 5½ feet. When the "weight" comes on the coal the work of hand mining is comparatively easy. Permissible explosives are used exclusively in this mine and give satisfactory results in shooting up the bottom bench, and only light charges are required to loosen the top bench, often only two short holes being necessary to shoot the top mass loose in a 24-foot room. Miners working in these rooms gain a larger tonnage per box of explosives used than the average for the regular machine districts of the mine.

QUANTITY OF EXPLOSIVE USED.

The amount of "snubbing" determines to a large extent the success obtained with permissible explosives. When the coal is well snubbed to a depth of 3 feet, the mass loosened by the shot tends to fall over, with the result that the coal is won easily and without special danger. Less explosive is used and the grade of coal obtained is better than without snubbing.

In the mines in the No. 6 coal, the charge when three shots are fired per room is usually three and one-half to four sticks, or 1½ to 2 pounds, for the center or "buster" shot, and three sticks, or 1½ pounds, for each of the rib shots. Practice varies somewhat in the different mines; in some as many as five sticks are used in the center shot, with four in each of the rib shots. One instance was noted in which six sticks were used for the center shot and five for each of the rib shots. This is in excess of the limit charge of 1½ pounds which the Bureau of Mines has determined as the maximum safe loading charge per hole; a greater number of holes should be employed with less explosive per hole. Sometimes four holes are drilled to meet these conditions.

The usual practice is to use a slightly heavier charge for the "buster" or center shot than for the rib holes. In some mines a number of the miners used three sticks of explosive per hole, includ-

ing the center shot. In such instances, however, the amount used was sufficient to break the coal down in the center.

In one mine in the No. 5 coal, the practice of using a lighter charge for the center shot was found to give good results. Two sticks of explosive were fired in the center shot and three in the rib shots. The center shot caused the coal merely to sag, tending to break the

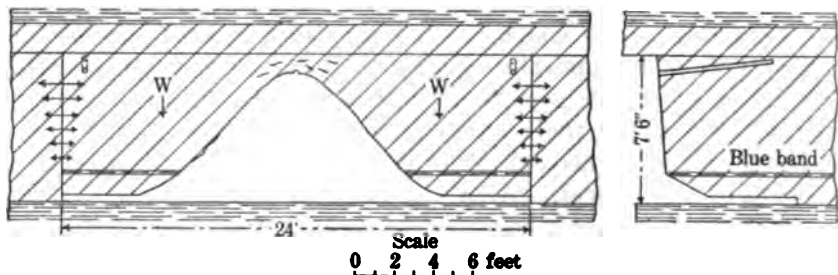


FIGURE 11.—Diagram showing effects of charging center shot heavier than the rib shots, showing tendency of remaining coal to break from ribs after partial removal of center support.

mass free from the ribs. The effects of these two methods are shown in figures 11 and 12.

In figure 11, the arrows at either rib indicate the tension thrown on the coal at the rib by the weight of the mass W , which, through the partial removal of the center support by the "buster" shot, acts as a cantilever to break the remaining coal free from the ribs. As

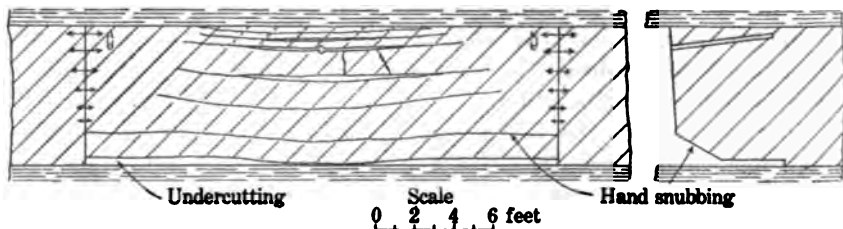


FIGURE 12.—Diagram showing lines of stress when coal is undercut and sagging in the middle from the discharge of a light center shot.

the coal does not have much tensile strength, the weight of the partly supported mass undoubtedly helps to break the coal free when the rib shots are fired.

The reason for using a heavier charge for the "buster" shot is to give two loose ends for the rib shots. The "buster" shot loosens a somewhat circular mass at the center of the face, which on account of the snubbing tends to fall forward. When the rib shots are fired, there is a side thrust, and the coal from these shots tends to sink down on the coal loosened by the "buster" or center shot.

When the center shot is fired light, as shown in figure 12, the effect is different. When such a shot is fired, the full length of the coal across the face acts as a partly supported beam, which tends to sag in the middle. The burden on the center shot when the coal has been undercut to a depth of 6 feet is much less than on the rib shots. The fact that the center shot is charged light and merely loosens the coal at the center but does not break it tends to cause a greater tension on the upper part of the rib, which aids the action of the explosive in the corner or rib shots. It is necessary that the rib shots shall be shearing shots, and with thick coal this makes a heavy burden on the shot. With the center sagging and tending to pull the coal from the ribs, the rib shots have less work to do and throw the coal slightly forward. Much, of course, is dependent on the character of the coal and its resistance to shearing action.

Table 3 gives records of shots with permissible explosives in typical mines in Franklin and Saline Counties, with data on the method of placing, drilling, and loading the hole, dimensions of hole, amount of explosive used, and coal produced.

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced.

MINES IN FRANKLIN COUNTY.

MINE A.

Place where shots were fired.	Width of room or entry.	Height of coal.	Depth of undercutting.	Snubbing.		Position of holes.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse. ^b	Depth of stemming.	Total sticks of explosive used.	Total cars of coal produced.	Total tons of coal produced.	Total pounds of explosive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.												
Room work. ^c																	
Entry 7 S. off 4 E. S.:	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>			<i>Tons. Cent.</i>		
Room 11.....	27 0	7 10	6 0	2 0	1 9	L	5 10 5 9 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10	1 3 1 3 1 3 1 1 1 1 1 1 1 1 1 1 1 1 1 1	2 0 2 3 2 3 2 3 2 3 2 3 2 3 2 3 2 3 2 3	34	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0	104	12	50 0	5.9	212
Room 19.....	27 6	7 3	6 0	2 0	1 8	L	5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10	1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1 1	2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0	44	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0	104	11	46 0	5.9	196
Room 21.....	29 1	7 7	6 0	2 6	1 6	L	5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10	1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0	2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0	44	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0	12	13	50 12	6.75	187
Room 22.....	27 9	7 8	6 0	2 6	1 6	L	5 9 5 9 5 9 5 9 5 9 5 9 5 9 5 9 5 9 5 9	1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0	2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0	44	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	2 10 2 10 2 10 2 10 2 10 2 10 2 10 2 10 2 10 2 10	12	12	49 0	6.75	181
Room 23.....	25 0	7 9	5 11	2 0	1 6	L	6 0 6 0 6 0 6 0 6 0 6 0 6 0 6 0 6 0 6 0	1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0	2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0	44	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0	12	12	48 18	6.75	181
Room 24.....	24 0	7 6	6 0	2 0	1 6	L	5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10 5 10	1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0 1 0	2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0	44	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	2 10 2 10 2 10 2 10 2 10 2 10 2 10 2 10 2 10 2 10	12	13	52 15	6.75	196
Total for 6 rooms.....													69	73	297 5	38.80	1,151
Average ^d	26 8	7 7	6 0	2 2	1 7		5 10	1 1	2 3	3.8	8	2 10	114	12	49 11	6.47	192
Entry 8 S. off 4 E. S.:																	
Room 24.....	28 0	7 7	6 11	3 0	1 6	L	7 0 7 0 7 0 7 0 7 0 7 0 7 0 7 0 7 0 7 0	0 11 0 11 0 11 0 11 0 11 0 11 0 11 0 11 0 11 0 11	2 3 2 3 2 3 2 3 2 3 2 3 2 3 2 3 2 3 2 3	34	8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E. 8 E.	3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0 3 0	94	14	58 0	5.3	274

Room 16.....	27 6	7 10	6 10	2 8	1 0	L. C.	6 6 6 7 6 8 6 10	1 3 1 3 1 3 1 6	2 2 2 2 2 2 2 2	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 8 2 8 2 8 2 8	3 8 3 8 3 8 3 8	10 10 10 10	14	55 0	5.6	246
Room 14.....	29 0	8 4	7 0	2 8	1 6	L. C.	6 10 6 10 6 9 6 9	1 0 1 0 1 0 1 0	2 0 2 0 3 0 3 0	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 0 2 0 2 0 2 0	3 0 3 0 3 0 3 0	10 10 10 10	15	63 0	5.6	281
Room 9.....	27 9	8 2	7 0	2 6	1 6	L. C.	6 9 6 9 6 9 6 9	0 11 0 11 0 11 0 11	3 0 3 0 3 0 3 0	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 10 2 10 2 10 2 10	3 10 3 10 3 10 3 10	10 10 10 10	14	56 8	5.6	232
Room 6.....	26 0	8 0	7 0	2 4	1 6	L. C.	6 10 6 10 6 10 6 10	1 0 1 0 1 0 1 0	2 0 2 0 2 0 2 0	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 6 2 6 2 6 2 6	3 6 3 6 3 6 3 6	10 10 10 10	13	52 0	5.6	232
Total for 5 rooms														49½	70	284 8	27.7	1,285
Average	27 8	8 0	6 11	2 8	1 6		6 9	1 0	2 3	3.3	8	2 10	3 10	9.9	14	56 18	5.5	257
Entry 9 S. off 4 E. S.:																		
Room 2.....	27 6	7 6	5 8	1 4	2 0	L. C.	5 8 5 8 5 8 5 8	1 0 1 0 1 0 1 0	2 6 2 6 2 6 2 6	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 6 2 6 2 6 2 6	3 6 3 6 3 6 3 6	9½	12	49 4	5.3	232
Room 3.....	26 0	7 11	5 8	1 6	2 2	L. C.	5 6 5 6 5 6 5 6	1 0 1 0 1 0 1 0	2 6 2 6 2 6 2 6	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 8 2 8 2 8 2 8	3 8 3 8 3 8 3 8	9½	14	56 14	5.3	267
Entry 10 S. off 4 E. S.:																		
Room 4.....	28 6	8 1	5 8	1 6	1 6	L. C.	5 8 5 8 5 8 5 8	1 2 1 2 1 2 1 2	3 3 3 3 3 3 3 3	34 34 34 34	8 E. 8 E. 8 E. 8 E.	2 10 2 10 2 10 2 10	3 10 3 10 3 10 3 10	10	11	45 2	5.6	201
Room 1.....	27 0	8 2	5 8	2 6	1 6	L. C.	5 9 5 9 5 9 5 9	1 0 1 0 1 0 1 0	2 8 2 8 2 8 2 8	34 34 34 34	8 E. 8 E. 8 E. 8 E.	3 0 3 0 3 0 3 0	3 0 3 0 3 0 3 0	9½	12	49 4	5.3	232
Total for 4 rooms.....														39½	49	200 4	21.5	932
Average	27 3	7 11	5 8	1 9	1 10		5 8	1 1	2 8	3½	8	2 9	3 9	9.6	12½	50 1	5.4	238
Entry 8 S. E.:																		
Room 19.....	26 0	8 0	6 6	4 0	3 6	L. C.	6 6 6 6 6 6 6 6	0 11 0 11 0 11 0 11	2 0 2 0 2 0 2 0	34 34 34 34	8 E. 8 E. 8 E. 8 E.	4 0 4 0 4 0 4 0	5 0 5 0 5 0 5 0	10½	14	58 0	5.9	246
Room 20.....	25 6	8 0	6 6	3 0	2 6	L. C.	6 6 6 6 6 6 6 6	1 2 1 2 1 2 1 2	2 0 2 0 2 0 2 0	4 4 4 4	8 E. 8 E. 8 E. 8 E.	4 0 4 0 4 0 4 0	5 0 5 0 5 0 5 0	10	13	54 10	5.6	248

a L signifies left; C center; and R, right.

b L for E signifies that electric detonator was used.

c Shot fired by electric battery. No electric detonator.

d "blue bird" 1 inch ½ inch thick. Face overhangs slightly, top projecting 1 to 2 feet. One round of shots per room.

e With 6-foot machine.

f With 7-foot machine.

Holes slope upward toward roof, back of hole being within 2 to 4 inches of the top. Parting

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced—Continued.

MINES IN FRANKLIN COUNTY—Continued.
MINE A—Continued.

Place where shots were fired.	Width of room or en- try.	Height of coal.	Depth of undercut- ing.	Snubbing.		Position of hole.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse.	Depth of stemming.	Total sticks of explo- sive used.	Total cars of coal pro- duced.	Total tons of coal pro- duced.	Total pounds of explo- sive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.												
Room work—Continued.																	
Entry 8 S. S. E.—Continued.	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>			<i>Tons. Cent.</i>		
Room 19 a.....	26 3	8 2	6 6	3 6	2 6	L. C. R.	6 6 6 6 6 6	0 11 1 0 0 11	2 0 2 0 2 0	3½ 3 4	8 E. 8 E. 8 E.	4 0 4 0 4 0	10½	14	58 0	5.9	246
Total for 3 rooms.....													31	41	170 10	17.4	735
Average.....	25 11	8 1	6 6	3 6	2 10		6 6	1 0	2 0	3½	8	4 0	10½	13½	56 17	5.8	246
Entry 8 N. N. W.:																	
Rooms 24 and 25.....	27 0	8 0	5 6	3 0	2 0	L. C. R.	5 4 5 3 5 4	1 0 1 2 1 0	2 0 2 6 2 0	4 3½ 4	8 E. 8 E. 8 E.	3 0 3 0 3 0	11½	13	53 0	6.5	204
	27 3	8 0	5 6	3 0	2 0	L. C. R.	5 0 5 4 5 4	1 2 1 4 1 4	2 0 2 0 2 0	4 4 4	8 E. 8 E. 8 E.	3 0 3 0 3 0	12	13	54 5	6.75	201
	27 0	8 5	5 6	3 0	2 0	L. C. R.	5 4 5 4 5 4	1 4 1 6 1 2	2 0 2 0 2 0	3½ 3½ 3½	8 E. 8 E. 8 E.	3 0 3 0 3 0	10½	14	54 0	5.9	229
Total for 3 rooms.....													34	40	161 5	19.15	634
Average.....	27 1	8 1	5 6	3 0	2 0		5 3	1 2	2 1	3½	8	3 0	11½	13½	53 15	6.38	211
Grand total for 21 rooms.....													222	273	1,113 12	124.55	4,737
Grand average, 21 rooms, representa- live of room work in Mine A.....	27 0	7 11	6 2	2 6	1 10		6 0	1 1	2 3	3.5	8	3 0	10½	13	53 1	5.9	226

MINE B.															
Room work. ^b															
Entry 10 S. E., off 4 E. S., average.....	11 2	6 10	5 6	2 0	$\left\{ \begin{array}{l} L. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 3 \\ 5\ 11 \\ 5\ 2 \\ 5\ 10 \end{array} \right\}$	$\left\{ \begin{array}{l} 0\ 11 \\ 1\ 0 \\ 2\ 0 \\ 1\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 8 \\ 1\ 0 \\ 2\ 0 \\ 1\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 24 \\ 44 \\ 34 \\ 4 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 8 \\ 2\ 8 \\ 2\ 8 \\ 2\ 8 \end{array} \right\}$	$\left\{ \begin{array}{l} 8 \\ 8 \\ 8 \\ 8 \end{array} \right\}$	$\left\{ \begin{array}{l} 5 \\ 5 \\ 5 \\ 5 \end{array} \right\}$	$\left\{ \begin{array}{l} 17\ 18 \\ 18\ 0 \\ 4\ 5 \\ 100 \end{array} \right\}$		
Entry 9 S. E., off 4 E. S., average.....	11 8	7 0	6 0	2 0	$\left\{ \begin{array}{l} L. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 10 \\ 5\ 10 \\ 5\ 10 \\ 5\ 10 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 1\ 0 \\ 1\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 2\ 0 \\ 2\ 0 \\ 2\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 4 \\ 4 \\ 4 \\ 4 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 8 \\ 2\ 8 \\ 2\ 8 \\ 2\ 8 \end{array} \right\}$	$\left\{ \begin{array}{l} 8 \\ 8 \\ 8 \\ 8 \end{array} \right\}$	$\left\{ \begin{array}{l} 5 \\ 5 \\ 5 \\ 5 \end{array} \right\}$	$\left\{ \begin{array}{l} 17\ 18 \\ 18\ 0 \\ 4\ 5 \\ 100 \end{array} \right\}$		
Total for 2 entries.....											16	10	35 18 9.0		
Average for entry work in Mine A.....	11 5	6 11	5 9	2 3	2 0	5 6	1 0	1 5	4	8	2 8	5	17 19 4.5		
Room work. ^b															
Room 19, entry 8 S. off 8 W. off main south entry.	29 6	7 9	6 9	2 2	1 7	$\left\{ \begin{array}{l} L. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 6\ 0 \\ 5\ 10 \\ 6\ 2 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 0\ 11 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 2\ 0 \\ 2\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 3 \\ 3 \\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 3\ 0 \\ 3\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 9 \\ 9 \\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 17 \\ 17 \\ 17 \end{array} \right\}$	$\left\{ \begin{array}{l} 54\ 10 \\ 4\ 5 \\ 303 \end{array} \right\}$	
Entry 7 S. off 8 W. off main south entry:															
Room 19.....	28 0	7 10	6 10	2 2	1 6	$\left\{ \begin{array}{l} L. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 6\ 3 \\ 6\ 2 \\ 6\ 4 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 2 \\ 1\ 3 \\ 1\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 9 \\ 3\ 0 \\ 3\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 3 \\ 3 \\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 3\ 0 \\ 3\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 9 \\ 9 \\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 18 \\ 18 \\ 18 \end{array} \right\}$	$\left\{ \begin{array}{l} 56\ 0 \\ 4\ 5 \\ 311 \end{array} \right\}$	
Room 16.....	24 0	7 8	6 9	2 2	1 6	$\left\{ \begin{array}{l} L. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 6\ 3 \\ 6\ 3 \\ 6\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 1 \\ 1\ 1 \\ 1\ 1 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 3 \\ 2\ 3 \\ 2\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 4 \\ 4 \\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 3\ 0 \\ 3\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 10 \\ 10 \\ 10 \end{array} \right\}$	$\left\{ \begin{array}{l} 17 \\ 17 \\ 17 \end{array} \right\}$	$\left\{ \begin{array}{l} 51\ 15 \\ 5\ 0 \\ 259 \end{array} \right\}$	
Total for three rooms.....															
Average c.....	27 2	7 9	6 9	2 1	1 6		6 2	1 1	2 2	3	6	3 0	94	52	162 5 14.0
Entry 6 S. off 10 W. off main south:															
Room 32.....	22 0	7 8	5 3	2 4	1 6	$\left\{ \begin{array}{l} L. \\ C. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 4\ 9 \\ 4\ 8 \\ 4\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 2 \\ 1\ 1 \\ 1\ 1 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 2\ 2 \\ 1\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 3 \\ 3 \\ 24 \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 6 \\ 5\ 3 \\ 5\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 9 \\ 9 \\ 94 \end{array} \right\}$	$\left\{ \begin{array}{l} 12 \\ 11 \\ 11 \end{array} \right\}$	$\left\{ \begin{array}{l} 35\ 10 \\ 4\ 5 \\ 200 \end{array} \right\}$	
Room 30.....	21 0	7 6	5 6	2 6	1 6	$\left\{ \begin{array}{l} L. \\ C. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 3 \\ 5\ 3 \\ 5\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 1\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 1\ 6 \\ 2\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 3 \\ 34 \\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 3 \\ 5\ 3 \\ 5\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 84 \\ 84 \\ 94 \end{array} \right\}$	$\left\{ \begin{array}{l} 11 \\ 12 \\ 12 \end{array} \right\}$	$\left\{ \begin{array}{l} 34\ 0 \\ 4\ 35 \\ 184 \end{array} \right\}$	
Room 31.....	21 6	7 6	5 6	2 6	1 6	$\left\{ \begin{array}{l} L. \\ C. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 3 \\ 5\ 3 \\ 4\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 1\ 2 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 6 \\ 2\ 0 \\ 2\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 34 \\ 3 \\ 24 \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 6 \\ 5\ 3 \\ 5\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 94 \\ 84 \\ 74 \end{array} \right\}$	$\left\{ \begin{array}{l} 12 \\ 11 \\ 11 \end{array} \right\}$	$\left\{ \begin{array}{l} 35\ 0 \\ 4\ 75 \\ 220 \end{array} \right\}$	
Room 30, entry 5 S. off 10 W. off main south.	22 0	7 5	5 0	2 0	1 7	$\left\{ \begin{array}{l} L. \\ C. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 4\ 8 \\ 4\ 8 \\ 4\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 1\ 1 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 6 \\ 2\ 0 \\ 1\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 24 \\ 24 \\ 24 \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 3 \\ 5\ 3 \\ 5\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 74 \\ 74 \\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 11 \\ 11 \\ 11 \end{array} \right\}$	$\left\{ \begin{array}{l} 33\ 0 \\ 3\ 75 \\ 189 \end{array} \right\}$	
Entry 3 N.:.....															
Room 19.....	24 0	7 6	5 0	2 0	1 8	$\left\{ \begin{array}{l} L. \\ C. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 0 \\ 5\ 0 \\ 5\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 1\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 11 \\ 2\ 2 \\ 1\ 8 \end{array} \right\}$	$\left\{ \begin{array}{l} 3 \\ 3 \\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 6 \\ 5\ 6 \\ 5\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 9 \\ 9 \\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 11 \\ 11 \\ 11 \end{array} \right\}$	$\left\{ \begin{array}{l} 34\ 2 \\ 4\ 5 \\ 189 \end{array} \right\}$	
Room 20.....	24 3	7 6	5 0	2 0	1 10	$\left\{ \begin{array}{l} L. \\ C. \\ R. \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 0 \\ 5\ 0 \\ 5\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 1\ 0 \\ 1\ 0 \\ 1\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 2\ 0 \\ 1\ 8 \\ 2\ 0 \end{array} \right\}$	$\left\{ \begin{array}{l} 3 \\ 3 \\ 3 \end{array} \right\}$	$\left\{ \begin{array}{l} 5\ 6 \\ 5\ 6 \\ 5\ 6 \end{array} \right\}$	$\left\{ \begin{array}{l} 9 \\ 9 \\ 9 \end{array} \right\}$	$\left\{ \begin{array}{l} 10 \\ 10 \\ 10 \end{array} \right\}$	$\left\{ \begin{array}{l} 34\ 0 \\ 4\ 5 \\ 189 \end{array} \right\}$	

^a Data taken one week after above data for this room was taken.

^b Holes incline toward roof about 1 inch per foot.

^c Seven-foot machine and wide rooms.

Data show that the "buster" shot often produced one-half of the coal. Face overhangs slightly.

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced—Continued.

MINES IN FRANKLIN COUNTY—Continued.

MINE B—Continued.

Place where shots were fired.	Width of room or entry.	Height of coal.	Depth of undercutting.	Shooting.		Position of hole.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse.	Depth of stemming.	Total sticks of explosive used.	Total cars of coal produced.	Total tons of coal produced.	Total pounds of explosive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.												
	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>L.</i> <i>C.</i> <i>R.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>			<i>Tons. Cwt.</i>		
Room work—Continued.																	
Entry 3 N.—Continued.																	
Room 21.....	24 6	7 6	5 1	1 8	1 9	L. C. R.	5 0 5 0 5 0	1 0 1 0 1 0	1 8 1 8 2 0	3 3 3	5 6 5 4 5 6	3 0 3 0 3 0	9	12	25 10	4.5	197
Total for seven rooms.....													61½	79	241 2	30.75	1,376
Average a.....	22 9	7 6	5 2	2 2	1 7		5 0	1 0	1 11	2.9	5 4	2 9	8.8	11.3	34 9	4.40	197
Average for room work in mine B.....	24 1	7 7	5 8	2 2	1 7		5 4	1 1	2 0	3	5 7	2 10	8.95	13	40 7	4.5	235
Room 17, entry 6 S. off 8 E. S.....	17 0	7 8	5 0	2 6	2 10	L. R.	4 6 4 8	1 5 1 7	2 0 2 0	3½ 3½	5½ 5½	2½ 2½	7	8	28 8	3.5	167
Narrow work (entries). ^b																	
Entry 9 W. off main south.....	12 0	6 8	6 8	1 0	2 0	L. R.	6 8 6 11	0 11 0 11	1 6 1 8	5 5	6 0 6 0	3 0 3 0	9	7	21 2	4.5	117
Entry 8 S. off 10 W. off main south.....	11 8	7 0	6 8	1 0	2 0	L. R.	6 6 6 6	0 11 1 0	2 0 2 0	5 5	6 0 4 6	3 0 3 0	9	7	21 5	4.5	118
Entry 2 S. off 13 E. S.....	12 6	7 4	5 4	3 0	2 6	L. R.	5 2 5 3	1 0 1 0	2 0 2 0	5 5	4 6 4 6	3 0 3 0	9	6	19 0	4.5	106
Entry 2 S. off 12 E. S.....	12 6	7 4	5 6	2 6	2 4	L. R.	5 0 5 4	1 0 1 6	2 0 2 0	5 5	4 6 4 6	3 0 3 0	9	7	22 0	4.5	122
Entry 13 E. S.....	12 0	7 4	5 4	2 1	2 0	L. R.	5 0 5 0	1 6 1 6	1 10 1 10	5 5	4 6 4 6	3 0 3 0	9	6	19 10	4.5	108
Entry 4 S. off 8 E. S.....	12 0	7 0	5 0	2 0	2 2	L. R.	5 2 5 0	1 2 1 2	1 7 1 6	3 3	4 6 4 6	3 0 3 0	7	6	18 5	3.5	130
Do.....	12 0	7 1	5 0	2 0	2 2	L. R.	5 0 5 0	1 2 1 2	1 6 1 7	3 3	4 6 4 6	3 0 3 0	7	6	18 0	3.5	129

Entry 5 S. off 8 E. S.	13 8	7 5	5 0	2 11	2 10	L. R.	4 8	1 0	2 0	3	5 6	2 4	7	6	17 13	3.5	126
Do.	13 8	7 5	5 0	3 0	2 10	L. R.	4 8	1 0	2 1	3	5 6	2 4	7	6	18 0	3.5	129
Total for nine entries							4 6	1 0	2 0	4	5 6						
Average for entry work in Mine B.	12 4	7 2	5 6	2 2	2 4		5 4	1 1	1 10	4	5 2	2 6	8.1	6.3	19 8	4.1	131

MINE C.

<i>Room work.</i> ^c																	
Entry 2 S. W., off No. 4 stub:																	
Room 8.	28 0	7 9	5 6	2 6	1 6	L. C. R.	5 0	1 0	1 6	4	6 0	2 6	13		48 0	6.7	100
Room 9.	28 5	7 8	5 6	2 5	1 6	L. C. R.	5 0	1 0	1 6	5	4 0	2 6					
Entry 2 S. W. off No. 6 stub:																	
Room 1.	28 0	7 6	5 6	2 6	1 6	L. C. R.	5 0	1 6	2 0	4	6 0	3 0	12½	13	48 10	6.7	103
Room 2.	27 8	7 8	5 6	2 6	1 6	L. C. R.	5 0	1 3	2 0	4	5 10	3 0					
Entry 4 N. W. off No. 2 stub:																	
Room 19.	27 6	7 6	5 6	2 0	1 6	L. C. R.	5 0	1 3	3 0	4½	5 6	2 6	14	12	40 15	7.2	141
Room 20.	28 2	7 6	5 6	2 0	1 6	L. C. R.	5 1	1 0	2 0	4½	6 0	2 6					
Total for six rooms.							5 1	1 0	2 0	4½	5 0	2 0					
Average for room work in Mine C.	28 0	7 7	5 6	2 4	1 6		5 0	1 2	2 0	4.4	5 5	2 7	13	77	264 10	40.6	943
<i>Entry work.</i>																	
Entry 5 S. off 2 N. E.	12 2	7 6	5 6	2 6	1 6	L. R.	5 0	1 0	1 6	4	6 0	2 6	9	6	19 0	4.6	103
Entry 6 S. off 2 N. E.	12 0	7 6	5 6	2 4	1 6	L. R.	5 0	1 0	1 6	4	5 0	2 6	9	6	18 10	4.6	101
Entry 1 N. off No. 3 stub off 2 S. W.	12 0	7 2	5 6	2 0	1 6	L. R.	5 1	1 0	1 6	5	6 0	2 0	9	6	18 10	4.6	101
Entry 2 N. off No. 3 stub off 2 S. W.	12 0	7 3	5 6	2 0	1 6	L. R.	5 2	1 0	1 6	4	5 6	2 6	8½	6	18 15	4.4	107
Total for four entries							5 1	1 0	1 6	4½	5 6	2 6					
Average for entry work in Mine C.	12 0	7 4	5 6	2 2	1 6		5 1	1 0	1 6	4.4	5 7	2 3	8.9	24	74 15	18.2	413

^a Short-wall machine, rooms of average width. ^b Average for mine. ^c Holes drilled on angle to strike top parting at back of hole. "Blue band" $\frac{3}{4}$ to 1 inch thick.

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced—Continued.

MINES IN FRANKLIN COUNTY—Continued.

MINE D.^a

Place where shots were fired.	Width of room or entry.	Height of coal.	Depth of undercut.	Snaubbing.		Position of hole.	Depth of hole.		Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse.	Depth of stemming.	Total sticks of explosive used.	Total cars of coal produced.	Total tons of coal produced.	Total pounds of explosive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.		Ft. ins.	Ft. ins.				Ft. ins.	Ft. ins.					
Room 10, entry 5 NE., off panel 6.	23 0	7 5	6 0	3 0	1 11	C.R.	5 9 5 10 6 0	1 0 0 10 1 0	1 6 2 2 2 0	24	24	5 8 5 8 5 8	2 10 2 10 2 10	9	12	36 1	4.6	196
Room 6, entry 8 NE., off panel 2.	22 6	7 11	6 0	2 9	1 7	C.R.	6 0 6 0 6 1	1 0 1 5 1 1	2 0 2 0 2 8	3	34	6 0 6 0 6 0	3 0 3 0 3 0	94	12	36 4	5.1	177
Entry 9 NE., off panel 1: Room 7.	23 0	6 7	6 0	2 4	1 4	C.R.	5 9 6 0 6 0	1 0 1 0 1 0	1 6 2 6 2 6	3	34	5 6 5 6 5 6	3 0 3 0 3 0	94	11	33 13	5.1	165
Room 3.	23 0	7 2	6 0	2 9	1 3	C.R.	5 9 6 0 6 0	1 0 1 0 1 0	2 6 2 6 2 1	3	34	5 6 5 6 5 6	3 0 3 0 3 0	94	11	33 10	4.8	174
Room 10.	23 0	7 5	6 0	2 4	1 6	C.R.	5 11 5 8 5 11	0 8 0 8 0 8	2 0 2 0 2 1	3	34	5 6 5 6 5 6	2 8 2 8 2 8	10	11	32 13	5.2	157
Entry 9 NE., off panel 2: Room 10.	24 0	7 2	6 0	2 9	1 7	C.R.	5 11 5 8 5 11	0 8 0 8 0 8	2 5 2 5 2 5	3	34	5 6 5 6 5 6	3 0 3 0 3 0	104	11	34 10	5.6	164
Room 12.	24 0	7 2	6 0	2 9	1 7	C.R.	5 11 5 8 5 11	0 8 0 8 0 8	2 5 2 5 2 5	3	34	5 6 5 6 5 6	3 0 3 0 3 0	9	11	33 10	4.7	178
Entry 1 NW., off panel 6: Room 5.	23 6	7 0	6 0	3 0	1 8	C.R.	5 9 5 9 5 10	0 11 0 11 0 11	2 6 2 6 2 6	24	24	5 6 5 6 5 6	3 0 3 0 3 0	94	12	35 15	4.8	186
Room 13.	23 6	7 0	6 0	3 0	1 8	C.R.	5 9 5 9 5 10	0 11 0 11 0 11	2 6 2 6 2 6	3	34	5 6 5 6 5 6	3 0 3 0 3 0	8	12	37 15	4.2	226
Entry 10 SE., panel 1: Room 5.	23 6	7 3	6 0	2 4	1 6	C.R.	5 11 5 6 5 6	0 8 1 0 1 0	2 4 2 4 1 0	3	34	5 6 5 6 5 6	2 10 2 10 2 10	9	13	39 11	4.8	206

Room 9.....	26 6	7 4	6 0	2 6	1 5	L. C.	5 10 5 4	1 7 1 9	1 9 2 1	5 6 5 6	3 0 3 0	9 4	14	42 13	4.8	232
Room 5, entry 10 SE., off panel 7.....	23 6	7 2	6 0	2 6	1 8	C. R.	5 2 5 2	0 10 2 0	2 3 4 4	5 0 5 0	3 0 3 0	10	12	37 11	5.2	180
Total for 12 rooms.....							5 2	0 10	3 3	5 0	3 0	113	142	433 06	58.9	2,220
Average for rooms of Mine D.....	23 7	7 3	6 0	2 8	1 7		5 10	1 0	2 2	5 8	2 11	9.4	11.8	36 02	4.9	185

MINE E.

Entry 3 R.....	12 0	7 3	6 0	5 9	3 0	L. R.	5 10 5 10	1 0 1 0	1 6 1 6	5 6 5 6	2 2 2 2	10 1/2	5	19	5.25	90
Entry 4 R.....	12 0	7 3	6 0	5 9	3 0	L. R.	5 9 5 9	1 0 1 0	1 6 1 6	5 6 5 6	2 2 2 2	10 1/2	5	20	5.25	96
Entry 4 R.....	12 0	7 3	6 0	5 9	3 0	L. R.	5 10 5 10	0 10 0 10	1 6 1 6	5 6 5 6	2 2 2 2	10 1/2	5	19 10	5.25	98
Entry 1 L., off 10 WS.....	12 2	7 6	6 0	5 6	3 6	L. R.	5 6 5 6	1 0 1 0	1 6 1 6	5 6 5 6	2 2 2 2	12	5	20	6.0	83
Entry 2 L., off 10 WS.....	12 1	7 6	6 0	5 6	3 6	L. R.	5 6 5 6	1 0 1 0	1 6 1 6	5 6 5 6	2 2 2 2	12	5	21	6.0	86
Total for 5 entries.....												55 1/2	25	99 10	27.75	449
Average for narrow work at Mine E.....	12 1	7 4	6 0	5 8	3 2		5 8	1 0	1 6	5 7	2	11	5	19 18	5.55	90
Entry A: Room 10.....	29 0	10 0	5 6	5 3	3 0	L. R.	5 0 5 0	1 3 1 3	1 8 1 8	6 6 6 6	4 6 4 6	14	16	64	7.0	229
Room 11.....	29 2	10 1	5 5	5 4	2 11	C. R.	5 0 5 0	1 4 1 4	1 8 2 0	6 4 6 4	4 4 4 4	15	16	64 15	7.5	216
Room 12.....	28 10	9 11	5 7	5 5	3	C. R.	5 2 5 2	1 3 1 3	2 0 1 8	6 4 6 4	4 4 3 6	14	16	65	7.0	232
Room 15.....	30 0	9 6	5 8	5 4	3	C. R.	5 1 5 2	1 1 1 2	2 0 2 0	5 6 5 6	3 6 3 6	13	15	60 10	6.5	233
Total for 4 rooms.....							5 2	1 2	2 0	5 6	3 6	56	63	254 05	28.0	910
Average for wide work in rooms.....	29 3	9 11	5 7	5 4	3 0		5 1	1 3	1 10	5 10	4 0	14	16	63 11	7.0	228

^a Series of shots with 3 different brands of permissible explosives. All holes were parallel with rib and inclined to roof $1\frac{1}{2}$ inches per foot.

^b Including 4 1/2 sticks of explosive in 2 snubbing shots.

^c Including 6 sticks of explosive in 2 snubbing shots.

^d Where some snubbing shots are fired.

^e Including 6 sticks in 3 snubbing shots.

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced—Continued.

MINES IN FRANKLIN COUNTY—Continued.

* MINE E—Continued.

Place where shots were fired.	Width of room or en- try.	Height of coal.	Depth of undercut- ting.	Snubbing.		Position of hole.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse.	Depth of stemming.	Total sticks of explo- sive used.	Total cars of coal pro- duced.	Total tons of coal pro- duced.	Total pounds of explo- sive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.												
Work in 84-foot coal.																	
Entry 7 R.:	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>			<i>Tons. Cwt.</i>		
Room 13.....	27	8 6	5 6	3 0	2 0	L.....	5 0	1 0	2 6	4	5	3 6	13	12	55	6.5	212
Room 14.....	27 4	8 6	5 6	3 0	2 0	C.....	5 0	1 0	2 6	4	5	3 6	13	12	54 18	6.5	211
Room 16.....	26 8	8 8	5 6	3 0	2 0	L.....	5 2	1 1	2 6	4	5	3 6	13	12	54 10	6.5	210
Total for 3 entries.....							5 2	1 2	2 0	4	5	3 6	39	36	164 08	19.5	633
Average for work in 84-foot coal.....	27 0	8 7	5 6	3 0	2 0		5 1	1 1	2 4	4.3	4 8	2 10	13	12	54 16	6.5	211
Grand average for room work in Mine E.....	28 3	9 4	5 6	4 4	2 7		5 1	1 2	2 0	3.3	5 4	3 6	13.6	14	59 16	6.8	220

Room work.

Entry 13 off 8 E.S.:

Room 1.....	27 0	6 6	6 6	2 0	1 6	L.C. R.	6 0 6 0 6 0 6 0 6 0 6 0	1 3 1 3 2 0 1 3 2 0 1 3	2 0 3 4 2 0 3 4 2 0 3 4	3 3 3 3 3 3	6 0 6 0 6 0 6 0 6 0 6 0	5 0 5 0 5 0 5 0 5 0 5 0	9 4 9 4 9 4 9 4 9 4 9 4	12 12 12 12 12 12	44 43 43 49 48 49	2 10 10 5 0 10	4.75 4.75 4.75 4.5 4.5 5.5	232 220 220 274 267 224	
Room 2.....	27 2	6 5	6 6	2 0	1 6	L.C. R.	6 0 6 0 6 0 6 0 6 0 6 0	1 3 1 3 2 0 1 3 2 0 1 3	2 0 3 4 2 0 3 4 2 0 3 4	3 3 3 3 3 3	6 0 6 0 6 0 6 0 6 0 6 0	5 0 5 0 5 0 5 0 5 0 5 0	9 4 9 4 9 4 9 4 9 4 9 4	12 12 12 12 12 12	43 43 43 49 48 49	10 10 10 5 0 10	4.75 4.75 4.75 4.5 4.5 5.5	220 220 220 274 267 224	
Entry 8 E.S.:																			
Room 1.....	27 0	7 0	6 6	2 6	1 6	L.C. R.	6 0 6 0 6 0 6 0 6 0 6 0	1 6 1 6 1 6 1 6 1 6 1 6	1 6 1 6 1 6 1 6 1 6 1 6	3 4 3 4 3 4 3 4 3 4 3 4	6 0 6 0 6 0 6 0 6 0 6 0	4 0 4 0 4 0 4 0 4 0 4 0	9 9 9 9 9 9	12 12 12 12 12 12	49 49 49 49 49 49	5 5 5 5 5 5	4.5 4.5 4.5 4.5 4.5 4.5	274 274 274 274 274 274	
Room 2.....	26 9	7 0	6 6	2 6	1 6	L.C. R.	6 0 6 0 6 0 6 0 6 0 6 0	1 6 1 6 1 6 1 6 1 6 1 6	1 7 1 7 1 7 1 7 1 7 1 7	3 4 3 4 3 4 3 4 3 4 3 4	6 0 6 0 6 0 6 0 6 0 6 0	4 0 4 0 4 0 4 0 4 0 4 0	9 9 9 9 9 9	12 12 12 12 12 12	48 48 48 48 48 48	0 0 0 0 0 0	4.5 4.5 4.5 4.5 4.5 4.5	267 267 267 267 267 267	
Entry 1 E. off 2 N.E.:																			
Room 1.....	27 4	7 1	6 6	2 0	1 0	L.C. R.	6 2 6 0 6 2 6 0 6 0 6 0	1 6 1 6 1 6 1 6 1 6 1 6	1 6 1 6 1 6 1 6 1 6 1 6	4 3 4 3 4 3	6 6 6 0 6 2 6 0 6 0 6 0	4 3 4 3 4 3 4 3 4 3 4 3	11 11 11 11 11 11	12 12 12 12 12 12	49 49 49 49 49 49	5 5 5 5 5 5	5.5 5.5 5.5 5.5 5.5 5.5	224 224 224 224 224 224	
Room 2.....	27 0	7 0	6 6	2 0	1 3	L.C. R.	6 0 6 0 6 0 6 0 6 0 6 0	1 6 1 6 1 6 1 6 1 6 1 6	1 8 1 8 1 8 1 8 1 8 1 8	4 4 4 4 4 4	6 6 6 0 6 2 6 0 6 0 6 0	4 0 4 0 4 0 4 0 4 0 4 0	11 11 11 11 11 11	12 12 12 12 12 12	49 49 49 49 49 49	10 10 10 10 10 10	5.5 5.5 5.5 5.5 5.5 5.5	226 226 226 226 226 226	
Total for six rooms.....														59	72	283	12 29.5	1.481	
Average.....	27 1	6 10	6 6	2 2	1 5		6 0	1 5	1 8	3 3	6 1	4 5	9.8	12	47	05	4.9	242	

Entries.^d

Entry 1 S. off 4 E.S.:

Entry 2 S. off 4 E.S.....	17 10	6 6	6 5	2 6	1 6	L.C. R.	6 0 6 0 6 0 6 0 6 0 6 0	1 3 1 3 1 6 1 6 1 6 1 6	1 6 3 4 3 4 3 4 3 4 3 4	6 3 6 0 6 3 6 0 6 0 6 0	4 0 4 0 4 0 4 0 4 0 4 0	10 10 10 10 10 10	8 8 8 8 8 8	20 20 20 20 20 20	16 16 16 16 16 16	57 57 57 57 57 57	10 10 10 10 10 10	288 288 288 288 288 288	5.0 5.0 5.0 5.0 5.0 5.0	145 145 145 143 143 144
Total.....																				
Average.....	17 11	6 6	6 5½	2 6	1 6		6 0	1 3	1 6	3½	6 1	4 0	10	8	28	15	5.0	144		

^a No snubbing shots fired.
^b Holes parallel with rib and sloping up toward roof 1 1/2 inches per foot.
^c A. Saline County mine. Shots are good average of the mine.
^d Wide entries. Production practically proportional to width.

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced—Continued.

MINES IN FRANKLIN COUNTY—Continued.

* MINE E—Continued.

Place where shots were fired.	Width of room or entry.	Height of coal.	Depth of undercutting.	Snubbling.		Position of hole.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse.	Depth of stemming.	Total sticks of explosive used.	Total cars of coal produced.	Total tons of coal produced.	Total pounds of explosive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.												
Work in 8½-foot coal.																	
Entry 7 R.:	Ft. ins.	Ft. ins.	Ft. ins.	Ft. ins.	Ft. ins.	R C R C R C R C	Ft. ins.	Ft. ins.	Ft. ins.		Ft. ins.	Ft. ins.			Tons. Cwt.		
Room 13.....	27	8 6	5 6	3 0	2 0		5 0 5 0 5 0 5 0 5 0 5 0 5 0 5 0	5 1 5 1 5 1 5 1 5 1 5 1 5 1 5 1	2 6 2 6 2 6 2 6 2 6 2 6 2 6 2 6	4 5 4 4 4 4 4 4	5 4 5 4 5 4 5 4	3 3 3 3 3 3 3 3	13	12	55	6.5	212
Room 14.....	27 4	8 6	5 6	3 0	2 0		5 5 5 5 5 5 5 5 5 5 5 5 5 5 5 5	5 1 5 1 5 1 5 1 5 1 5 1 5 1 5 1	3 5 3 5 3 5 3 5 3 5 3 5 3 5 3 5	4 5 4 4 4 4 4 4	5 4 5 4 5 4 5 4	3 3 3 3 3 3 3 3	13	12	54 13	6.5	211
Room 16.....	26 8	8 8	5 6	3 0	2 0	5 2 5 2 5 2 5 2 5 2 5 2 5 2 5 2	5 1 5 1 5 1 5 1 5 1 5 1 5 1 5 1	2 0 2 0 2 0 2 0 2 0 2 0 2 0 2 0	4 5 4 4 4 4 4 4	5 4 5 4 5 4 5 4	3 3 3 3 3 3 3 3	13	12	54 10	6.5	210	
Total for 3 entries.....													39	36	164 08	19.5	633
Average for work in 8½-foot coal.....	27 0	8 7	5 6	3 0	2 0		5 1	1 1	2 4	4.3	4 8	2 10	13	12	54 16	6.5	211
Grand average for room work in Mine E.....	28 3	9 4	5 6	4 4	2 7		5 1	1 2	2 0	3.3	5 4	3 6	13.6	14	59 16	6.8	220

Room work.

Entry 13 off 8 E.:

Room 1.....	27 0	6 6	6 6	2 0	1 6	LOG-LOC	6 0 6 0 6 0 6 0	1 3 1 3 1 3 1 3	2 0 2 0 2 0 2 0	3 3 ^a 3 3 ^a	6 0 6 0 6 0 6 0	5 0 5 0 5 0 5 0	9 ^b 9 ^b 9 ^b 9 ^b	12 12 12 12	44 43 43 43	2 10 10 10	4.75 4.75 4.75 4.75	232 229 229 229
Room 2.....	27 2	6 5	6 6	2 0	1 6	LOG-LOC	6 0 6 0 6 0 6 0	1 3 1 3 1 3 1 3	2 0 2 0 2 0 2 0	3 3 ^a 3 3 ^a	6 0 6 0 6 0 6 0	5 0 5 0 5 0 5 0	9 ^b 9 ^b 9 ^b 9 ^b	12 12 12 12	43 43 43 43	10 10 10 10	4.75 4.75 4.75 4.75	229 229 229 229
Entry 8 E S.:																		
Room 1.....	27 0	7 0	6 6	2 6	1 6	LOG-LOC	6 0 6 0 6 0 6 0	1 6 1 6 1 6 1 6	1 6 1 6 1 6 1 6	3 ^a 3 ^a 3 ^a 3 ^a	6 0 6 0 6 0 6 0	4 0 4 0 4 0 4 0	9 9 9 9	12 12 12 12	49 49 49 49	5 5 5 5	4.5 4.5 4.5 4.5	274 274 274 274
Room 2.....	26 9	7 0	6 6	2 6	1 6	LOG-LOC	6 0 6 0 6 0 6 0	1 6 1 6 1 6 1 6	1 7 1 7 1 7 1 7	3 ^a 3 ^a 3 ^a 3 ^a	6 0 6 0 6 0 6 0	4 0 4 0 4 0 4 0	9 9 9 9	12 12 12 12	48 48 48 48	0 0 0 0	4.5 4.5 4.5 4.5	267 267 267 267
Entry 1 E. off 2 NE.:																		
Room 1.....	27 4	7 1	6 6	2 0	1 0	LOG-LOC	6 2 6 0 6 2 6 0	1 6 1 6 1 6 1 6	1 6 1 6 1 6 1 6	4 3 4 3	6 6 6 0 6 6 6 0	4 3 4 3 4 3 4 3	11 11 11 11	12 12 12 12	49 49 49 49	5 5 5 5	5.5 5.5 5.5 5.5	224 224 224 224
Room 2.....	27 0	7 0	6 6	2 0	1 3	LOG-LOC	6 0 6 0 6 0 6 0	1 6 1 6 1 6 1 6	1 8 1 8 1 8 1 8	4 4 4 4	6 6 6 0 6 6 6 0	4 0 4 0 4 0 4 0	11 11 11 11	12 12 12 12	49 49 49 49	10 10 10 10	5.5 5.5 5.5 5.5	226 226 226 226
Total for six rooms.....														59	72	283	12	29.5
Average.....	27 1	6 10	6 6	2 2	1 5		6 0	1 5	1 8	3 3	6 1	4 5	9.8	12	47	05	4.9	242

Entries.^d

Entry 1 S. off 4 E S.....	18 0	6 6	6 6	2 6	1 6	{ L. C. R. }	6 0 6 0 6 0 6 0	1 3 1 3 1 3 1 3	1 6 1 6 1 6 1 6	3 ^a 3 ^a 3 ^a 3 ^a	6 3 6 0 6 3 6 0	4 0 4 0 4 0 4 0	10 10 10 10	8 8 8 8	20 20 20 20	0 0 0 0	5.0 5.0 5.0 5.0	145 145 145 145	
Entry 2 S. off 4 E S.....	17 10	6 6	6 6	2 6	1 6	{ L. C. R. }	6 0 6 0 6 0 6 0	1 2 1 2 1 2 1 2	1 6 1 6 1 6 1 6	3 ^a 3 ^a 3 ^a 3 ^a	6 0 6 0 6 0 6 0	4 0 4 0 4 0 4 0	10 10 10 10	8 8 8 8	28 28 28 28	10 10 10 10	5.0 5.0 5.0 5.0	143 143 143 143	
Total.....							6 0	1 3	1 6					20	16	57	10	10.0	288
Average.....	17 11	6 6	6 6 ^b	2 6	1 6		6 0	1 3	1 6	3 ^a	6 1	4 0	10	8	28	15	5.0	144	

^a No snubbing shots fired.
^b Holes parallel with rib and sloping up toward roof 1½ inches per foot.
^c A Saline County mine. Shots are good average of the mine.
^d Wide entries. Production practically proportional to width.

TABLE 3.—Records of shots with permissible explosives in typical mines, showing data on method of placing, drilling, and loading the holes, amount of explosive used, and coal produced—Continued.

MINES IN FRANKLIN COUNTY—Continued.

*MINE E—Continued.

Place where shots were fired.	Width of room or entry.	Height of coal.	Depth of undercutting.	Snubbling.		Position of hole.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used.	Length of fuse.	Depth of stemming.	Total sticks of explosive used.	Total cars of coal produced.	Total tons of coal produced.	Total pounds of explosive used.	Tons of coal produced per 25-pound box of explosive.
				Depth.	Height.												
Work in 8½-foot coal.																	
Entry 7 R.:	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>		<i>Ft. ins.</i>	<i>Ft. ins.</i>	<i>Ft. ins.</i>			<i>Ft. ins., Ft. ins.</i>			<i>Tons. Crd.</i>		
Room 13.....	27	8 6	5 6	3 0	2 0	L. C.R.	5 0 5 0 5 0	1 0 1 0 1 0	2 6 2 6 2 6	4 5 5	5 4 5	3 6 3 6 3 6	13	12	55	6.5	212
Room 14.....	27 4	8 6	5 6	3 0	2 0	L. C.R.	5 1 5 1 5 1	1 1 1 1 1 1	2 5 2 5 2 5	5 5 5	5 4 5	6 6 6	13	12	54 18	6.5	211
Room 16.....	26 8	8 8	5 6	3 0	2 0	L. C.R.	5 2 5 2 5 2	1 2 1 2 1 2	2 0 2 0 2 0	4 4 4	5 4 5	3 6 3 6 3 6	13	12	54 10	6.5	210
Total for 3 entries.....													39	36	164 08	19.5	638
Average for work in 8½-foot coal.....	27 0	8 7	5 6	3 0	2 0		5 1	1 1	2 4	4.3	4 8	2 10	13	12	54 16	6.5	211
Grand average for room work in Mine E.....	26 3	9 4	5 6	4 4	2 7		5 1	1 2	2 0	3.3	5 4	3 6	13.6	14	59 16	6.8	220

Room work.

Entry 13 off S E.

Room 1.....	27 0	6 6	6 6	2 0	1 6	L. C.	6 0	1 3	2 0	3	6 0	5 0	12	44	2	4.75	232
Room 2.....	27 2	6 6	6 6	2 0	1 6	L. C.	6 0	1 3	2 0	3	6 0	5 0	12	43	10	4.75	229
Entry 8 E S.																	
Room 1.....	27 0	7 0	6 6	2 6	1 6	L. C.	6 0	1 6	1 6	3	6 0	4 0	9	49	5	4.5	274
Room 2.....	26 9	7 0	6 6	2 6	1 6	L. C.	6 0	1 6	1 6	3	6 0	4 0	9	48	0	4.5	267
Entry 1 E. off 2 N E.																	
Room 1.....	27 4	7 1	6 6	2 0	1 0	L. C.	6 2	1 6	1 6	4	6 6	4 2	11	49	5	5.5	224
Room 2.....	27 0	7 0	6 6	2 0	1 3	L. C.	6 0	1 5	1 7	4	6 0	4 0	11	49	10	5.5	226
Total for six rooms.....							6 1	1 6	1 8	4	6 6	4 0		283	12	29.5	1,451
Average.....	27 1	6 10	6 6	2 2	1 5		6 0	1 5	1 8	3 3	6 1	4 5	9.8	47	05	4.9	242

Entries.^d

Entry 1 S. off 4 E S.....	18 0	6 6	6 6	2 6	1 6	L. C.	6 0	1 3	1 6	3	6 0	4 0	10	20	0	5.0	145
Entry 2 S. off 4 E S.....	17 10	6 6	6 6	2 6	1 6	L. C.	6 0	1 3	1 6	3	6 0	4 0	10	28	10	5.0	143
Total.....							6 0	1 2	1 6	3	6 0	4 0		57	10	10.0	288
Average.....	17 11	6 6	6 6	2 6	1 6		6 0	1 3	1 6	3	6 1	4 0	10	28	15	5.0	144

^a No snubbing shots fired.
^b Holes parallel with rib and sloping up toward roof 11 inches per foot.
^c A Saline County mine. Shots are good average of the mine.
^d Wide entries. Production practically proportional to width.

TABLE 4.—Summary of data shown in Table 3, including cost of explosive per ton of coal produced.

ROOM WORK.

Mine.	Number of rooms or entries observed.	Average width of room or entry.	Average height of coal.	Average depth of undercutting.	Squabbling.		Average number of shots per room or entry.	Average depth of holes.	Average distance of holes from roof.	Average distance of rib holes from rib.	Average number of sticks of explosive per hole.	Average length of fuse per shot. ^a	Average depth of stem-ming.	Average total number of sticks of explosive per fall.	Average number of cars of coal produced per fall.	Average tonnage produced per fall.	Average number of tons produced per shot.	Average weight of explosive used per fall.	Average number of tons produced per 25-pound box of explosive.	Average cost of explosive per ton of coal produced. ^b	Average cost of fuse and detonator per ton of coal produced. ^c
					Average depth.	Average height.															
A.....	21	27 0	7 11	6 2	2 6	1 10	3	6 0	1 1	2 3	3 5	8 3	3 0	10 6	13	33 01	17 14	5 9	226	\$0.011	\$0.0017
B.....	10	24 1	7 7	5 8	2 2	1 7	3	5 4	1 1	2 0	2 0	5 7	2 10	8 95	13	43 07	13 09	4 5	226	\$0.011	.0028
C.....	6	28 0	7 7	5 6	2 2	1 6	3	5 0	1 1	2 0	4 4	5 5	2 7	13	13	40 08	14 02	6 8	157	.016	.0026
D.....	12	23 7	7 3	6 0	2 8	1 7	3	5 10	1 0	2 0	2 1	5 8	2 11	9 4	11 8	36 02	12 01	4 9	185	.013	.0032
E.....	7	28 3	9 4	5 6	2 4	2 2	3	5 1	1 1	2 0	2 3	5 4	3 6	13 6	14	59 16	11 19	6 8	220	.011	.0031
F.....	6	27 1	6 10	6 6	2 2	1 5	3	6 0	1 1	2 0	2 3	5 1	4	9 8	12	47 05	15 15	4 9	243	.010	.0026
Total for six mines	62															278 19	85 0	33 8	1,265		
Average.....		26 4	7 9	5 11	2 8	1 9	3	5 6	1 2	2 0	3 4	6 7	3 2	10 9	12 8	46 10	14 03	5 6	209	.013	.0027

ENTRY WORK.^c

A.....	2	11 5	6 11	5 9	2 3	2 0	2	5 6	1 0	1 5	4	8 3	2 6	8 1	5	17 19	9 0	4 5	100	\$0.026	\$0.0083
B.....	9	12 4	7 2	5 6	2 2	1 4	2	5 4	1 1	1 10	4 4	5 2	2 3	8 9	6 0	16 08	9 14	4 1	121	.020	.0037
C.....	4	12 0	7 4	5 6	2 2	1 6	2	5 1	1 0	1 6	4 4	5 7	2 3	8 9	6 0	18 14	9 07	4 55	108	.024	.0041
E.....	5	12 1	7 4	6 0	2 3	2 2	4	5 8	1 0	1 6	3	5 7	2 0	11	5	19 18	5 0	5 55	90	.027	.0076
Total for four mines	20															75 19	33 01	18 7	414		
Average.....		12 0	7 2	5 8	3 1	2 3	2 5	5 5	1 0	1 7	3 85	5 5	2 4	9 0	5 6	19 00	8 05	4 7	104	.024	.0047

^a Letter E signifies that electric detonator was used.^b With normal prices for fuse and detonators.^c Franklin County mines only.

PREPARATION OF THE PRIMING CARTRIDGE.

When detonators or blasting caps and fuse are used in preparing the priming cartridge, the following points are to be considered:

The detonator contains a highly sensitive substance, known as fulminate of mercury, and it should be handled with care.

The successful firing of the shot largely depends on careful and proper arrangement of the fuse and detonator.

The ingredients in fuse readily absorb moisture, also a part of the powder train may be lost from the open end, and therefore an inch or two of fuse should always be cut off the end of the roll before it is used.

The fuse should be cut off squarely with a sharp knife, the end being held in an upright position to prevent loss of powder. The end of the fuse should then be pushed gently up into the detonator until it touches the fulminate. It should not be twisted in the detonator, as the friction might be sufficient to explode the detonator.

The fulminate readily absorbs moisture; therefore the detonator should be kept in a moisture-proof container until it is to be used. The detonator should not be blown into with the breath, as moisture may thus be introduced.

The detonator should be crimped on the fuse with a proper crimper, providing a practically water-tight connection. Plate VI shows an improved type of crimper and the method of properly crimping the detonator to the fuse. The very common practice of crimping the detonator to the fuse with a knife is inefficient; crimping with the teeth is a dangerous practice. A satisfactory crimper may be bought for about 25 cents each, and as it is needed each day should be regarded as an indispensable part of the miner's equipment. It is to be regretted that so few are in use.

In Illinois mines the usual custom of preparing the primer is to punch two holes diagonally, one entirely through the cartridge of explosive and the other part way, usually with a wooden skewer. The fuse is then laced through the open diagonal hole in the cartridge^a and the detonator end pushed into the other hole diagonally into the cartridge so that the detonator lies near the center of the cartridge, as shown in A, figure 13. The electric detonator is usually inserted in a hole punched diagonally into the side or end of the cartridge, and the wires are given a half hitch around the stick, as shown in B and C, figure 13.

By these methods the priming cartridges are easily and quickly prepared, and there is apparently some advantage in having the stemming^b forced against the end of the explosive with less possi-

^a The Bureau of Mines does not recommend the lacing of fuse through the cartridge of explosive.

^b In the publications of the Bureau of Mines, the word "stemming" is used to designate the material used to confine an explosive in a drill hole, and the word "tamping" is used to designate the act of ramming the stemming into the hole.

bility of doubling back the fuse or wires and in preventing the detonator from being pulled from the cartridge during the tamping, as they serve to do. There are, however, several serious disadvantages in the use of these methods, and they are not regarded as the best practice for the following reasons: Care must be taken to see that the detonator lies near the center of the charge or the charge may not be detonated with full strength. If the explosive fills the drill hole tightly there is danger, when the cartridge is pushed into the hole, of breaking the insulation of the wires, or if fuse is used, the

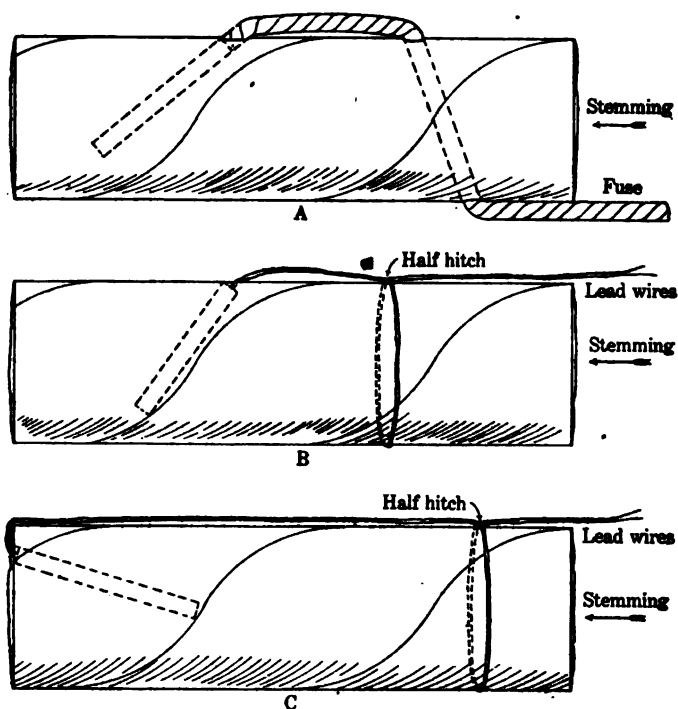


FIGURE 13.—Methods of arranging primers commonly used in the Illinois mines, not recommended. A, fuse laced through cartridge; B, electric detonator inserted in the side of cartridge and wires secured with half hitch; C, electric detonator inserted diagonally in end of cartridge and wires secured with half hitch.

fuse projecting through the cartridge may be damaged when being inserted. With the fuse laced through the cartridge there is always the possibility that the explosive may be ignited by "side spitting" of the fuse, start to burn, and thus prevent complete detonation. If tamping be done too vigorously, with the electric detonator arranged as shown, the cartridge containing the electric detonator may be so expanded that the half hitch in the wire may form a tight knot and possibly give a short circuit or cause the wires to break.

Practices that are sometimes followed in Illinois and are recommended as being more efficient than the others are shown in figure 14, in



A. IMPROVED TYPE OF CRIMPER,



B. CRIMPING DETONATOR ON FUSE.

which the detonator with fuse is inserted in the end or side, and the electric detonator in the end of the cartridge, and tied in position as shown. The first two or three inches of the tamping material should be inserted loosely against the charge so that the more forceful tamping will not disturb the position of the detonator nor subject it to shock. These methods require that the miner provide himself with string to tie the detonator in place, but this important part of each day's work should be carefully attended to and sufficient time taken to perform the work with proper care. A simple and excellent method of using an electric detonator is shown in figure 15.

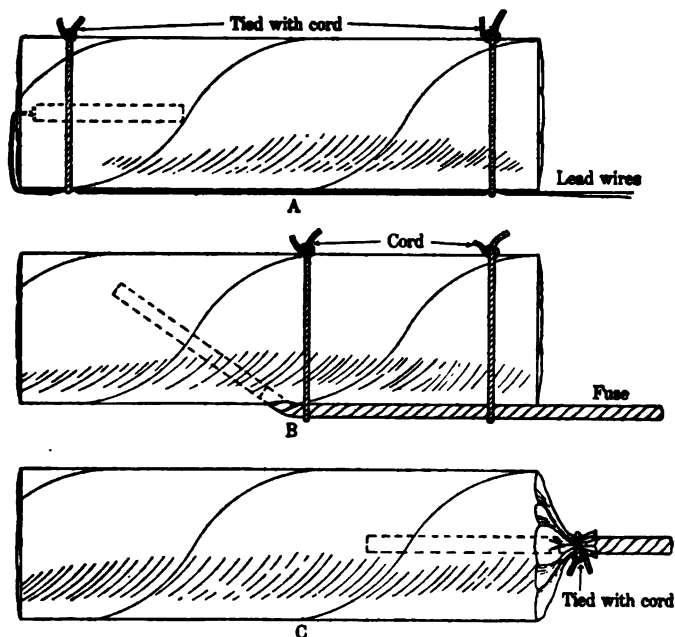


FIGURE 14.—Methods of arranging primers. A, electric detonator, wires tied with cord; B, fuse and detonator, detonator inserted in the side of cartridge and fuse tied to cartridge with cords; C, detonator inserted in end of cartridge and tied in with cord.

LOADING AND TAMPING THE HOLE.

Permissible explosives in cartridges $1\frac{1}{4}$ by 6 inches or $1\frac{1}{2}$ by 7 inches in size and weighing approximately one-half pound have been found to be the most satisfactory in Illinois and are used exclusively. The use of this diameter of cartridge, which is larger than ordinarily used in coal mining elsewhere, places a sufficient quantity of explosive at the back of the drill hole to do the required work, without the cartridges being broken. The miner usually scrapes all drill dust from the hole before he inserts the cartridges, so that no material can lodge between them and prevent close contact. The cartridges are placed in the drill hole in

their original wrappers, usually without the paper ends being torn off, and are pushed back with the tamping bar. Some miners place the primer, containing the cap, in last, others place it in the middle when using three sticks. It is usually the last or next to the last cartridge put in.

Clay stemming is generally used and tamping is accomplished by means of a heavy iron bar, copper tipped.^a The stemming is usually made up in paper cartridges, called "dummies," about 2 inches in diameter and 12 to 15 inches long. Generally speaking, 3 feet of stemming is sufficient for satisfactory results with the usual charge

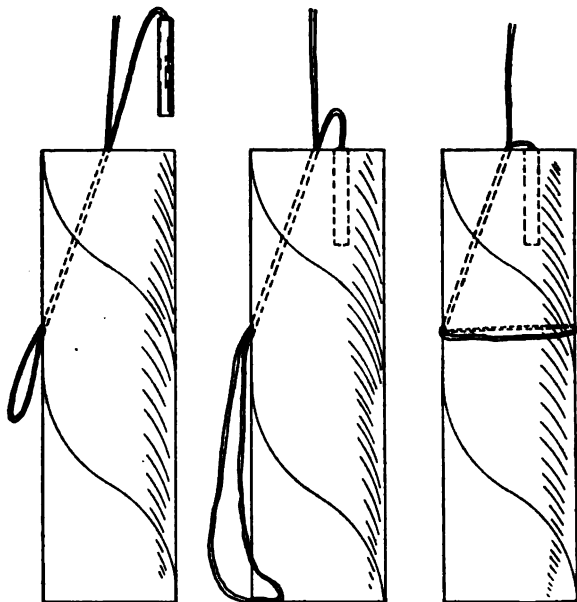


FIGURE 15.—Another method of inserting electric detonator in primer.

in a 6-foot hole, but sometimes the drill hole is directly above and parallel to a hard "sulphur" or "rock" band, and it takes more stemming to prevent the charge from blowing out. The amount of stemming should be sufficient not only for ordinary conditions but for extraordinary conditions, and therefore the stemming should always extend to the mouth of the drill hole. The State law in regard to tamping reads as follows:^b

Tamping.—Every blasting hole shall be tamped full from the explosive to the mouth of the hole, and no coal dust or any material that is inflammable or that may create a spark, whether the same shall be wet or dry, shall be used for tamping.

^a The Bureau of Mines recommends the use of a wooden tamping bar.

^b Coal-mining laws of Illinois, sec. 19 (i).

Although the law thus decrees that only noninflammable material shall be used as stemming, experience has shown that where such material can not be had conveniently some of the men will use "bug dust," or drill dust, unless close discipline is maintained, and some shot firers have no objections to its use. In every mine in the district clay is obtainable from the floor. Sometimes it is obtained with difficulty, but the practical miners know that by pouring some water or coffee from their dinner pails on a convenient spot the clay overnight becomes moist and breaks up easily; and thereafter it is a simple matter to obtain all necessary clay for tamping by occasional dampening. Where clay is not obtainable from the floor, no doubt the operator would be willing to send to convenient points in the mine suitable surface clay or dirt. Clay, in addition to being the more efficient material for holding the charge, tends to damp the flame in the event of a blown-out shot. The dangers of coal-dust stemming have been shown in tests with black blasting powder at the Pittsburgh experiment station of the Bureau of Mines; the use of coal-dust stemming lengthened the flame of a blown-out shot 50 to 60 feet.*

There is no defense for the use of coal dust for stemming. Several times when its use was called to the attention of the miners a majority of them strongly denounced the practice. A number of the State inspectors' reports recommend that the use of drillings be discontinued and that clay be used.

Care should be taken in tamping. The heavy iron bar commonly used should be abandoned, as it is dangerous, and a wooden tamping bar substituted. It is possible to strike a spark from a lump of pyrite, or "sulphur," with a metal rod, even of copper. The wooden tamping bar is much safer, is less liable to injure the fuse or detonator wires, and is sufficiently heavy to insure efficient tamping.

If wet holes are encountered, they are usually left uncharged and the shot firer does the charging and tamping; otherwise all shots are tamped by the miner. In the event that he quits work before the shot firer makes his rounds, the miner takes a piece of cartridge paper and marks on it the figure indicating the number of shots prepared in his room and posts it near the room mouth for the information of the shot firer. In some mines two brands of permissible explosives are used, and in a few instances both brands have been used in a single drill hole. This practice is not prevalent and has been done without understanding the danger of the practice, when a miner had but one or two sticks of a certain brand left and borrowed some of another brand from a fellow miner, or changed to a new brand. A fire may result from the use of two different brands in the same shot.

* Rice, G. S., The explosibility of coal dust: Bull. 20, Bureau of Mines, 1911, pp. 53-54.

METHODS OF FIRING SHOTS.

Relative to shot firing in Illinois mines, the State law in part reads:^a

In all mines in this State where coal is blasted, and where more than 2 pounds of powder is used for any one blast; and also, in all mines in this State where gas is generated in dangerous quantities, a sufficient number of practical, experienced miners, to be designated as shot firers, shall be employed by the company, and at its expense, whose duty it shall be to inspect and do all the firing of all blasts, prepared in a practical, workmanlike manner, in said mine or mines.

Shot firers are employed in all the mines using permissible explosives, from two to six shot firers being employed at each mine. They enter the mine at 1.30 to 2 p. m. and examine the shots to be fired, noting the place in the mine and the number of the shots. After inspection, the shot firers return to the shaft bottom. Later, after the miners have all left the mine, they fire the shots, completing their work in two and one-half to six hours. In the inspection of shots they work singly; when firing they work in pairs, this being in accordance with the working agreement between the operators and miners.^b Firing the three shots in a room at the same time is commonly practiced, the center, or "buster," shot having a shorter fuse than the two rib shots. When firing with fuse and detonator, the fuses are all lighted at the same time; with battery and electric detonator, the shots are fired one at a time.

As shown in Table 1, in all but two mines the shots are fired by fuse and No. 6 detonators. In two mines electric detonators and battery are used. In firing by electricity the shot firers carry about 110 feet of duplex No. 14 cable and a firing battery that can fire one to four holes. This system offers many advantages as regards safety and efficiency. Some of these advantages are as follows:

When firing is done by detonator and fuse, the interval of time between the exploding of the "buster" shot and the second shot is often not more than a second or two. The first shot will always stir up a certain amount of fine dust and in a gaseous mine may liberate some gas, and if the second shot contains an excessive charge of explosive and is poorly and insufficiently tamped, as sometimes happens in mines even under the closest inspection, a "windy" shot may result. With electric firing such a condition could not arise. There can be no "hang fires" and there are possibly fewer misfires, as the electric detonator gives better protection against wet holes, not being easily affected by moisture. In storage and use, the electric detonators are much more satisfactory from practically every point of view than fuse and detonators. The electric detonators are prepared ready for use, and can be tested before being used by means of

^a See pp. 97 to 99 for State law and working agreement between operators and miners relative to firing of shots.

^b See pp. 97-98.

a galvanometer; the objections to crimping are overcome; they are not so liable to be spilled or lost as the ordinary detonator; a better check can be made on the grade of detonators used and the use of detonators of proper strength more easily enforced; their use eliminates flame, as in lighting fuse or as in the "side spitting" of fuse, and provides for firing shots in a desired order. If the "buster" shot fails, the rib shots need not be fired. The method is safer and more satisfactory and conforms more fully with the requirements for the use of permissible explosives.

The cost of electric detonators with 8-foot iron-wire leads, at normal prices, is 3 cents to the miner. The cost of fuse is about one-half cent per foot; detonators cost 1 cent each. The average shot requires about 5 feet of fuse; hence the cost to the miner, with the electric detonators, is perhaps slightly less than when fuse is used.

Shot firers' wages cost the operator about \$0.0075 per ton of coal mined in four typical mines in Franklin County. For one Montgomery County mine the cost is about \$0.008 and for one Saline County mine \$0.0075. The cost per ton for shot firing in mines using battery and electric detonators is approximately the same as in mines employing shot firers, averaging between \$0.007 and \$0.0075 per ton of coal. The shot firers are paid for eight hours' work at a uniform rate of 58½ cents per hour.

NUMBER OF SHOTS FIRED.

The total number of shots fired per shift in the 19 mines listed in Table 1 is about 4,400; the daily average per mine is 231. The approximate number of shots fired nightly by each shot firer for "fuse and detonator" mines is 60; for "battery and electric detonator" mines, 56. As required by the State law, a notice is posted at each mine at the end of each shift showing the number of shots fired, the number not fired and reasons therefor, and the number of misfires. The form following is used:

SHOT FIRERS NOTICE.

(To be posted in a conspicuous place at the mine immediately after the completion of the work of firing shots—Section 2, Shot Firers Law.)

Name of owner or operator of mine: ——— *Coal Mining Co.* No.....

We, the undersigned shot firers, have this.....day of.....191..
fired.....shots in this mine. We have also refused to fire.....shots.

The shots we have refused to fire, and our reasons for not firing same, are as follows:

LOCATION.	REASONS FOR NOT FIRING SHOTS.

SHOTS FAILING TO EXPLODE.

LOCATION.	LOCATION.

.....Shot Firer.	Shot Firer.	Shot Firer.
.....Shot Firer.	Shot Firer.	Shot Firer.
.....Shot Firer.	Shot Firer.	Shot Firer.

From the records examined at the various mines it was evident that some shot firers required more strict adherence by the miners to rules than others. The reasons for refusals to fire shots at one mine during a period of two months were noted as follows: Six, not properly snubbed; four, "bug dust" not loaded out; four, crosscut in entry not through;^a two, car in the way.

MISFIRES.

The proportion of misfires in the mines is considerably less than 1 per cent. The records at one mine show only four misfires in a period of three months and the firing of upwards of 10,000 shots. At another mine three misfires occurred in 3,500 shots; at another, four in four months, or 25,000 shots. The usual impression is that rarely more than two misfires occur in one night at any one mine. Some data on misfires at two mines are given in the following tables:

^a The mining law states that not more than three shots shall be exploded at one shooting time ahead of the last open crosscut. See Appendix, p. 99.

Record from one mine for period of two months, detonators and fuse used:

Date.	Number of shots fired.	Misfires.	Remarks.
1915.			
January.....	4,821	5	104,206 tons of coal was produced during this period, or 8,000 tons for each misfire.
February.....	3,980	8	
Total.....	8,811	13	

Percentage of misfires, 0.15 per cent.

Data on shots fired during 20 months at one mine, using electric detonators and battery system.

Date.	Number of shots fired.	Number of misfires.	Date.	Number of shots fired.	Number of misfires.
1914.			1915.		
July.....	5,498	21	June.....	3,059	5
August.....	7,525	5	July.....	4,227	9
September.....	7,073	10	August.....	4,146	8
October.....	6,936	10	September.....	6,046	19
November.....	6,443	5	October.....	7,558	17
December.....	6,716	14	November.....	5,327	14
			December.....	7,497	19
1915.			1916.		
January.....	7,024	12	January.....	6,755	20
February.....	3,880	7	February.....	7,459	14
March.....	4,187	6			
April.....	3,823	5	Total.....	112,566	221
May.....	1,878	1			

Remarks.—Percentage of misfires, 0.2 per cent. As shots are fired one at a time by battery, this system affords an actual check on the number of misfires, and this record is much more accurate than records with fuse and detonator system. In one instance three shots misfired in one room the same night; in another, two. The following reasons for refusals to fire shots are given in the shot firers' records for this 20-month period: Top "working," 44; "bug dust" not loaded, 28; too far ahead of air, 16; car in way at face, 14; not snubbed, 8; place falling in, 6; too many shots ahead of air, 4; "buster" shot failed, 4; hole "on the solid," 3; retamped holes previously fired, 3; machine in way, 3; explosive in "buster" hole took fire and burned, remaining holes not fired, 1; detonator wire broke, 1; no primer made, 1.

PREVENTION OF MISFIRES.

The prevention of misfires is a matter of interest to the miner, both from safety and economy, and the subject should receive his careful consideration. The explosive with which he is supplied should be in good condition and he should take pains to keep it so. It is readily understood that with all the material in good condition and the shot properly arranged, misfires will seldom happen. As previously discussed, proper storage of fuse, detonators, and explosive, careful preparation of the primer, with use of crimper if fuse and detonator are used, and proper charging of the drill hole, are essential.

A few examples of causes of misfires investigated during the past year are as follows: A miner had been carrying detonators loose in his pockets. Possibly some tobacco or dust lodged on the fulminate of the detonator and prevented the "spit" of the fuse from reaching

and exploding it. Such practice is inefficient and dangerous. In another instance a shot that misfired had been charged two hours previously. The detonator had been crimped with a knife and moisture had seeped in. In another the fuse was broken during tamping.

Careless storing of fuse before it is used accounts for many misfires and hangfires, but a greater number result from carelessness in handling the fuse when it is inserted in the drill hole. The large copper-tipped iron tamping bars ordinarily used in Illinois mines are heavy, and the fuse is often crushed between the edge of the heavy bar and the sides of the drill hole. Rough handling may break the powder train or so damage the fuse that water can readily enter it if the hole or the stemming is damp, or the end of the fuse may lose part of the powder train and give a chance for misfires. The double-tape fuse is much better than cheaper grades because it will withstand more severe handling and is not so easily crushed.

Sometimes the end of the fuse, if damp from being exposed to the mine air or from lying on the damp bottom for a considerable time, can be inserted in the detonator only with difficulty, and the miner is unable to get it closer to the fulminate than about one-half inch. In that event the copper walls of the capsule containing the fulminate might absorb enough heat from the flame so that it would not ignite the fulminate. Fuse in good condition will have a "spit" at least $1\frac{1}{4}$ inches in length, but under the conditions mentioned—that is, damp fuse and the end of the fuse one-half inch from the fulminate composition—the "spit" may be reduced to a quarter of an inch or less. Even if the end which has been exposed to moisture is left for the shot firer, it may "spit" when lighted, burn for an inch or so, and go out, thus giving a missed shot.

In two different instances of shot failures in the same room, the stemming was blown out but the charge did not explode, indicating that old explosive or weak detonators had been used, as a box of detonators without a cover was found in this room.

An instance was noted in which fuse had been stored for a long period in a warm, poorly ventilated magazine, and the heat had caused the tar composition used in the manufacture of the fuse to run, giving a black color to the fuse. Much handling of fuse of this character might cause the tar composition to get in the powder train and cause a misfire.

COSTS OF PERMISSIBLE EXPLOSIVES PER TON OF COAL, AND SIZES OF COAL PRODUCED.

Table 1 shows that the highest average production of coal per 25-pound box of explosive for a period of one year in any of the mines listed, including all room and entry work, is 215 tons; the lowest

is 107 tons. As the mines listed include mines in various stages of development and others fully developed, a fair average can be obtained. The average production for 18 mines is 147 tons per 25-pound box of explosive. The average cost of explosive per ton of coal produced, with explosive selling at \$2.45 per box, is therefore $\$2.45 \div 147 = 1.7$ cents, the cost varying from the lowest figure, $\$2.45 \div 215 = 1.14$ cents, to the highest, $\$2.45 \div 107 = 2.3$ cents. To these figures must be added the cost of fuse and detonators per ton of coal, which averages 0.25 cent for fuse and 0.1 cent for detonators. These averages for fuse and detonators are obtained from data on five typical mines, as follows:

Data on number of detonators used and production at five mines for period of one year.

Mine.	Year ending—	Number of detonators used.	Number of tons produced.
No. 1.....	Mar. 31, 1915	36,300	420,612
No. 2.....	do	65,600	637,359
No. 3.....	Dec. 31, 1914	46,700	563,127
No. 4.....	do	21,300	291,351
No. 5.....	Mar. 31, 1915	50,300	528,930
Total.....		220,200	2,441,379

Production per detonator used.....tons..	11
Production per foot of fuse used.....do....	2
Average length of fuse per detonator used.....feet..	54
Average cost of detonators per ton of coal (detonators at 1 cent each).....	\$0.0010
Average cost of fuse per ton of coal.....	.0025
	.0035

Adding to this the average cost of explosive per ton of coal produced, 0.017 cent, gives an average total cost of \$0.0205.

In one mine using electric shot firing, during a period of 14 months ended June 30, 1915, 50,100 electric detonators were used. The production was 858,365 tons; the production per electric detonator used was 17 tons; and the average cost of electric detonators per ton of coal produced, \$0.0018. On the basis of these figures, the cost for electric detonators is \$0.0018, as compared with fuse and detonator at \$0.0035. The use of electric detonators is therefore cheaper for the miner.^a

The foregoing figures include general mine averages. Data on results of shots, taken at random from typical mines, in rooms and entries are given in Table 3. This table shows that the average cost of fuse and detonators per ton of coal in rooms with normal prices prevailing, was \$0.002; with electric detonators the cost was \$0.0017.

^a The cost of detonators and electric detonators increased about 50 per cent in 1916 through the increased demand for explosives.

In entry work the average cost for fuse and detonators at two mines was \$0.0039, and for electric detonators at one mine was \$0.0033. This table also shows that the average production was 209 tons of coal per 25-pound box of explosives in rooms, or the cost of explosives per ton of coal was 1.1 cents. In some rooms a production of 300 tons per 25-pound box of explosive was obtained. The average production in entries was 104 tons per 25-pound box, the explosive cost per ton of coal being 2.4 cents. It is felt that on the average the miner in rooms can easily obtain 200 tons of coal per 25-pound box of explosive at a cost of less than 1.5 cents per ton of coal produced, including fuse and detonators.

Additional data on the cost of explosives per ton of coal produced from a test with two brands of permissible explosives at one mine are as follows:

Results of tests of two permissible explosives at one mine.

Permissible explosive used.	Number of places shot.	Average depth of holes.	Explosive used.	Total cost.	Coal produced.	Cost per ton.	Coal produced per 25-pound box explosive.
		<i>Ft. in.</i>	<i>Pounds.</i>		<i>Tons.</i>		<i>Tons.</i>
No. 1.....	4	5 10	19.9	\$1.95	145.15	\$0.0134	182
No. 2.....	4	5 11	18.5	1.81	147.40	.0123	200
Average.....						.0128	191

The percentages of different sizes of coal produced with the use of permissible explosives at ten mines in Franklin county, the coal being screened over shaking screens with round holes, were as follows:

Sizes of coal produced with permissible explosives in 10 mines.

Mine No.	Lump, 6 inches in diameter.	Egg, 3 to 6 inches in diameter.	Nut, 2 to 3 inches in diameter.	Screenings, 2 inches in diameter.
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
1.....	18.22	19.77	10.96	51.05
2.....	22.08	20.40	12.52	45.00
3.....	18.00	22.15	10.50	49.35
4.....	22.00	14.00	16.00	48.00
5.....	17.00	23.00	12.00	48.00
6.....	27.34	20.29	11.91	40.46
7.....	24.14	20.67	12.60	42.59
8.....	22.05	21.62	11.86	44.47
9.....	23.44	20.43	9.28	46.85
10.....	21.73	17.42	10.65	50.20
Average.....	21.60	19.97	11.83	46.60

COMPARISONS WITH USE OF BLACK BLASTING POWDER IN ILLINOIS MINES.

COSTS OF EXPLOSIVE PER TON.

Comparative data on the cost of explosives per ton of coal and the sizes of coal produced from all the mines were not obtainable, owing to the fact that the different mines were using both permissible explosives and black blasting powder, or were shooting off the solid and also mining by machines at the same time. Some mines changed from shooting off the solid with black blasting powder to machine mining and permissible explosives; other mines have used permissible explosives exclusively and have no data on black blasting powder. Changes in methods of preparation of coal have also been made.

A few typical comparisons of the relative productive values of black blasting powder and permissible explosives are as follows:

Data showing comparative production of coal with equal amounts of black blasting powder and permissible explosive at one mine.

BLACK BLASTING POWDER.

Month.	Coal produced, tons.	Number of 25-pound kegs of black blasting powder used.	Tons of coal produced per 25-pound keg of black blasting powder.
October, 1908.....	38,748	430	90.10
November, 1908.....	30,270	363	83.38
Total.....	69,016	793	87.08

-- PERMISSIBLE EXPLOSIVE.

Month.	Coal produced, tons.	Number of 25-pound boxes of permissible explosive used.	Tons of coal produced per 25-pound box of permissible explosive.
October, 1910.....	49,706	286	173.79
November, 1910.....	48,612	274	177.41
Total.....	98,318	560	175.55

The above figures show a production of two tons to one in favor of permissible explosives with the use of equal amounts of explosive.

The average production per unit of explosive for a period of one year at five mines which changed from black blasting powder to permissible explosives is as follows:

Average production at five mines with black blasting powder and permissible explosives.

Mine No.	Tons previously produced per 25-pound keg of black blasting powder.	Tons produced per 25-pound box of permissible explosive.
1.....	90	130
2.....	100	202
3.....	85	128
4.....	68	107
5.....	131	215
Average.....	95	156
Ratio.....	1	1.64

Explosives cost per ton of coal produced with black blasting powder at \$1.75 per 25-pound keg..... \$0.0184

Explosives cost (including detonators) per ton of coal produced with permissible explosive at \$2.45 per 25-pound box..... .0167

The above data include mines in Franklin, Saline, and Montgomery Counties and are representative of the comparative production by black blasting powder and permissible explosives in mines in which the physical character of the coal differs considerably.

Data available from an early test in determining the price to miners for permissible explosives in Franklin County mines in 1910 are given as follows:

Data on shots with black blasting powder and a permissible explosive.

Place where shot was fired.		Shots with black blasting powder.			Shots with a permissible explosive.		
Entry.	Room.	Number of shots.	Quantity of explosive.	Coal produced.	Number of shots.	Quantity of explosive.	Coal produced.
			<i>Lbs. oz.</i>	<i>Tons.</i>		<i>Lbs. oz.</i>	<i>Tons.</i>
5 NW.....	5.....	3	4 13	31	3	3 14	22
4 SW.....	18.....	3	6 7	43	3	2 11½	32
4 SW.....	25.....	3	5 8	36	3	4 6½	38
2 NW.....	3.....	3	7 4	39	3	3 8	41
2 NW.....	4.....	3	6 10	38	3	3 13	40
Total.....		15	30 10	187	15	18 4	173
Average per shot.....		1	2 ⅔	12.5	1	1 8.5	11.5
Coal produced per pound of explosive.....				6.2			9.5
Tons of coal per 25-pounds of explosive.....				155.0			287.5

Ratio of production per pound of explosive: Black powder, 1; permissible explosive, 1.53. If the shots in room 5, entry 5 NW, in which the permissible explosive did not produce its full amount of coal, be eliminated, then for 12 shots in four rooms the relative production is—black powder 1, permissible explosive 1.75.

The records show that in three instances some work was necessary to obtain all the coal from the shots with the permissible explosive, but on the whole the results were satisfactory.

The results of a test at one mine with black powder and three brands of permissible explosives by the joint State powder commission were as follows:

Results of tests with black blasting powder and three brands of permissible explosives.

Explosive.	Number of places shot.	Quantity of explosive used.	Number of squibs used.	Length of fuse used.	Number of detonators used.	Total cost of explosive.	Amount of coal produced.	Cost per ton of coal.	Production per 25-pound box explosive.
		<i>Pounds.</i>		<i>Feet.</i>			<i>Tons.</i>		<i>Tons.</i>
Black blasting powder..	4	29.50	12	-----		\$2.08	191-3	\$0.0108	162
Permissible explosive No. 1.....	12	54.75	None.	233	35	6.83	525-	.0130	240
Permissible explosive No. 2.....	15	67.25	None.	240	42	8.21	624-	.0131	232
Permissible explosive No. 3.....	15	72.75	None.	250	44	8.82	629-	.0140	216

The average production with the three permissible explosives was 228 tons per 25-pound box. The ratio of production was: Black powder, 1; permissible explosive, 1.4. This ratio has been considerably improved since permissible explosives came into general use at this mine.

Data from the annual coal reports of the State mining board show an average production of 116 tons of coal per 25-pound keg of black blasting powder for a three-year period at the above mine, including rooms and entries. The average production per 25-pound box of permissible explosives is 215 tons. The ratio of production per unit of black blasting powder to permissible explosives is therefore 1 : 1.85.

A test by the powder commission at another mine gave the following results:

Results of tests at another mine.

Explosive and place.	Number of places shot.	Average length of hole.	Quantity of explosive used.	Length of fuse used.	Tons produced.	Cost per ton.
		<i>Ft. in.</i>	<i>Lb. oz.</i>	<i>Ft. in.</i>	<i>Tons, cwt.</i>	
Black blasting powder:						
Rooms.....	5	5 5	57 5	72 0	243 19	a \$0.0164
Entries.....	4	5 6	36 11	34 10	71 8	a .0370
Permissible explosive:						
Rooms.....	5	5 5	32 0	79 6	220 2	b .0152
Entries.....	4	5 8	16 15	44 10	62 15	c .0297

a Does not include cost of fuse.

b Add $\frac{1}{4}$ cent per ton for detonators.

c Add $\frac{1}{2}$ cent per ton for detonators.

Price of black blasting powder, \$1.75 per 25-pound keg; permissible explosive, \$2.45 per 25-pound box. Ratio of production with black blasting powder to production with permissible explosive: In rooms, 1 : 1.6; in entries, 1 : 1.9.

The figures of the test show that the cost for explosives per ton of coal produced in entries is practically double that in rooms. The previous average at this mine was 68 tons per 25-pound keg of black blasting powder. The present average with the permissible explosive used is 107 tons per 25-pound box. The permissible explosives, therefore, produce 1.6 times as much coal per unit as black blasting powder, which closely bears out the figures of the first test.

With black blasting powder, at \$1.75 per 25-pound keg, and permissible explosive, at \$2.45 per 25-pound box, permissible explosive is considerably cheaper for the miner on the basis of cost of explosives and detonators ^a per ton of coal produced.

Data by Andros ^b give for 43 machine mines in Illinois using black blasting powder, a powder cost of 2.1 cents per ton of coal; and for 34 mines in which shooting from the solid is practiced, a powder cost of 8.3 cents per ton of coal.

RELATIVE SIZES OF COAL.

A few typical comparisons of the sizes of coal produced, from data obtained at various mines, are as follows:

The sizes of coal produced in one mine for two consecutive months in 1912 with black blasting powder are as follows:

Production with black blasting powder for two months in 1914.

Size of coal.	Amount produced during—		Average.
	First month.	Second month.	
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
6-inch lump.....	20	19	19.5
3 to 6 inch egg.....	22	19	20.5
2 to 3 inch nut.....	7	14	10.5
2-inch screenings.....	51	48	49.5
Total.....	100	100	100.0

The production in the same mine for three consecutive months with permissible explosives in 1914 was as follows:

Production with permissible explosives for three months in 1914.

Size of coal.	Amount produced during—			Average.
	First month.	Second month.	Third month.	
	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>
6-inch lump.....	20	19	17	18.7
3 to 6 inch egg.....	24	26	24	24.7
2 to 3 inch nut.....	6	5	9	6.6
2-inch screenings.....	50	50	50	50.0
Total.....	100	100	100	100.0

^a The cost of detonators, as shown on p. 67, is approximately 0.1 cent per ton of coal produced, and is very low because a large production of coal is gained per shot. The average cost in some rooms is not more than 0.07 cent per ton of coal produced.

^b Andros, S. O., Coal Mining in Illinois, Bull. 13, Coal Mining Investigations, 1915, pp. 116, 122.

This shows that comparatively as good results are obtained with permissible explosives and that the slight decrease in the proportion of 6-inch lump is offset by increase in the egg sizes with little increase in screenings.

The results at another mine were as follows:

Relative sizes obtained at one mine.

	Lump, 6-inch.	Egg, 1½ to 6 inch.	Screen- ings, 1½-inch.
Previous average yield with black blasting powder.....	12.35	46.88	39.77
Yield with permissible explosives for week ending Aug. 15, 1914.....	12.49	47.94	39.57
Yield with permissible explosives for week ending Dec. 12, 1914.....	12.04	47.05	40.91
Yield with permissible explosives for period of Jan. 1-15, 1915.....	11.71	46.32	41.97
Average for 4 weeks with permissible explosives.....	11.99	46.91	41.10

This showing is favorable for permissible explosives. The results at another mine during the last year of using black blasting powder and the first year of using permissible explosives were as follows:

Comparative results for periods of one year each.

Size of coal.	Last year of using black powder.	First year of using per- missible.
	<i>Per cent.</i>	<i>Per cent.</i>
Lump, 6-inch.....	24.19	20.41
Egg, 3 to 6 inch.....	19.28	21.29
Nut, 2 to 3 inch.....	18.50	12.88
Screenings, 2-inch.....	40.03	45.42

The comparison above represents perhaps the maximum increase in screenings due directly to the use of permissible explosives. A small amount of pillar coal is included in the percentages for permissible explosives.

One of the latest mines in which the use of permissible explosives has been substituted for black blasting powder reports excellent results both in cost of explosive to the miner and sizes of coal produced. The cost of explosives to the miner has been reduced practically 50 per cent, and the proportion of larger sizes of coal has been increased. Records kept for a two-month period show a decrease of 3½ per cent in the amount of 1½-inch screenings.

It is generally agreed that the use of permissible explosives may give slightly more screenings than the use of black blasting powder, but the results obtained have been convincing proof that under favorable conditions the increase in proportion of screenings from using permissible explosives can be kept at a low figure. The increase at the majority of these mines ranges from nothing to 1 or 2 per cent, with a maximum of 5 per cent. As the miner is paid on the mine-

run basis, he has no special incentive to produce lump coal; hence these figures may be accepted as the results obtained from the use of permissible explosives under regular working conditions without an attempt to obtain a maximum amount of lump coal.

THE DEMAND FOR THE DIFFERENT SIZES.

The preparation of a great number of sizes of coal is necessary at Illinois mines to meet the market demands. Most of the mines where permissible explosives are used are equipped to load eight to ten sizes of coal. At one mine equipment was provided to load 17 sizes of coal, as shown in the following table:

Data on equipment for loading 17 sizes of coal.

Number of combination.	Track 1.		Track 2.		Track 3.		Track 4.	
	Kind of coal.	Size of coal.	Kind of coal.	Size of coal.	Kind of coal.	Size of coal.	Kind of coal.	Size of coal.
		<i>Inches.</i>		<i>Inches.</i>		<i>Inches.</i>		<i>Inches.</i>
1.....	Lump...	$\frac{1}{2}$					Screenings.	$\frac{1}{2}$
2.....	do...	2					do...	
3.....	do...	6	Egg...	2 by 6	Nut...	$\frac{1}{2}$ by 2	do...	
4.....	do...	$1\frac{1}{2}$			do...	$\frac{1}{2}$ by 2	do...	
5.....	do...	2			Nut...	$1\frac{1}{2}$ by 2	do...	
6.....	do...	6	Egg...	2 by 6	do...	$1\frac{1}{2}$ by 2	do...	
7.....	do...	3			do...	$1\frac{1}{2}$ by 3	do...	
8.....	do...	6	Egg...	3 by 6	do...	$1\frac{1}{2}$ by 3	do...	
9.....	do...	6	do...	$1\frac{1}{2}$ by 6			do...	
10.....	do...	2					do...	
11.....	do...	3			Nut...	2 by 3	do...	
12.....	do...	6	Egg...	3 by 6	do...	2 by 3	do...	
13.....	do...	3			Screenings.	3	do...	
14.....	do...	6	Egg...	3 by 6	do...	3		
15.....	do...	6	Mine run	6				

Very little run-of-mine coal is shipped from the mines. As noted on page 68 the proportion of lump coal and screenings produced at 10 of the mines under discussion averages 53.4 per cent lump and 46.6 per cent of 2-inch screenings. The demand for these sizes is variable. During a part of the year the markets require the larger sizes of coal, which command a higher price than screenings; therefore, it is of interest to the operator to produce at such times ~~and~~ a percentage as possible of these larger sizes. The greatest demand for lump coal probably comes in the late summer and in the fall and winter, with a consequent rise in prices during these periods. It is, however, necessary to produce screenings in the proportion noted above whenever lump coal is produced; therefore, the price of screenings tends to be lower during the periods when the price of lump coal is highest.

The demand for screenings, which comes largely from steam power plants, is more nearly constant during the whole year; consequently the mines, to meet the demand for screenings, must in the spring

and summer months produce lump coal for which there is little demand. This may necessitate for certain periods crushing of the lump coal produced. During the spring of 1915 one company crushed 400 tons of 6-inch lump daily for a time, and other companies are considering the installation of machinery for crushing coal during the spring and summer months. During the previous winter, however, some companies were forced to store their screenings in piles at the mines when the greatest demand was for the larger sizes; then when the demand for lump coal fell off and that for screenings continued steady, steam shovels were employed to load the screenings, while thousands of tons of lump loaded on cars waited for a market.

In this connection Weeks states ^a that—

Under present conditions (spring of 1915) it matters but little whether we are able to produce 20 or 40 per cent of screenings or 80 or 60 per cent of larger sizes; there arises daily the necessity to reduce prices on one or more sizes to prevent congestion at the mines.

The demand for small prepared coal and for screenings has been increasing year by year. This demand and the high price obtained for the nut sizes, which are taken from the screenings, as a domestic fuel is at some of these mines a factor that increases the relative value of the fine sizes, greatly prolongs the period in which it may pay to produce the smaller coal, and assists to maintain a better balance in the value of the whole product. Owing to these varying demands it is difficult to form a correct basis for computing how much a possible small increase in screenings from the use of permissible explosives might effect the average value of the mine output.

Taking the average price of lump coal over 2-inch screens as \$1.48 and the average price of 2-inch screenings as \$0.73, as representative of Franklin County market prices ^b for the years 1913 and 1914, the difference in value per ton is \$0.75, and consequently each increase of 1 per cent of screenings decreases the value of the product, in terms of mine-run coal, three-fourths of a cent per ton.

Although it is desired that the largest possible production of the larger sizes be maintained, the feeling prevails among the operators that the advantages of permissible explosives make their use economical even with a slight increase in the proportion of screenings. Hence the present problems do not include considerations of a return to the use of black blasting powder, but possible ways and means of improving the results obtained with permissible explosives. Such items are discussed in a following chapter.

^a Weeks, H. W., The influence of size on the cost of making steam: The Black Diamond, Mar. 20, 1915, p. 223.

^b Holbrook, E. A., Dry preparation of bituminous coal at Illinois mines: Bull. 88, Eng. Exp. Station, University of Illinois, 1916, p. 76.

FACTORS AFFECTING COST OF EXPLOSIVES PER TON OF COAL AND SIZES OF COAL PRODUCED.

There are a number of factors that influence the cost of explosives per ton of coal and the sizes of coal produced. It is difficult to explain the differences in the results obtained at the various mines, for these factors may vary in different parts of the same mine. In general they may be summarized as follows: (1) Physical structure of the coal and roof, (2) extent of undercutting and snubbing, (3) methods of use, (4) order of firing shots, and (5) width of workings.

PHYSICAL STRUCTURE OF THE COAL AND ROOF.

Conditions favorable and unfavorable to shooting down coal are usually indicated in the production of coal per unit of explosive used. The thickness of the bed, its cleat, friability, the nature of the roof, and impurities, such as shale or "sulphur" bands and irregularities, all affect blasting. Other conditions being equal, the thicker the coal the greater is the production per unit of explosive. The No. 6 bed has certain characteristics which tend to affect the ease of mining, such as the absence of definite cleat, and the variation presented in the three distinct benches—the brittle top coal, the somewhat stronger middle bench, the "blue band" parting, and the firm and hard bottom coal. The average production per unit of explosive in coals No. 5 and 6 is not in proportion to the thickness of the beds. In mining the No. 6 coal the top layer of coal is left for roof and the parting is not so free as the more pronounced roof separation of the No. 5 coal in Saline County. This is especially true where the roof is a light-colored shale, for then the parting is very distinct and the coal breaks away freely. The cleat in No. 5 coal in Saline County is probably more pronounced than in the No. 6 coal anywhere in the State. Consequently where No. 5 coal has its maximum thickness the production per unit of explosive is slightly higher than the average for No. 6 coal.

As to friability, these coals are regarded as being somewhat stronger and firmer than the average bituminous coals and they withstand shipment well. Tests to determine breakage due to handling tend to show that Illinois coals shot by permissible explosives are not affected materially by handling as compared with the same coals shot by black blasting powder. Especially in the egg and nut sizes these coals are firm and hard and suffer little loss from breakage in ordinary handling.

In certain mines in the No. 6 bed, where limestone closely overlies the coal with 2 to 8 feet of shale intervening, the top coal is not left up, but is often unusually hard and contains many bands of

"rock" and "sulphur." Such bands affect both the cost of mining and the sizes produced, because greater explosive force is required to break the coal and in the removal of impurities the coal undergoes further breakage. Often "sulphur" bands or "rolls" are encountered in drilling and the miner is compelled to shoot a short hole with possibly unsatisfactory results and at increased cost.

Plates VII and VIII show the No. 6 coal after being shot down with permissible explosives in typical mines in Franklin County.

UNDERCUTTING AND SNUBBING.

Table 1 (p. 22) shows that the average depth of undercutting by chain machines in the 18 mines listed is 6 feet, with a maximum of 7 feet and a minimum of 5 feet. The average depth of undercutting by puncher machines is 4.75 feet. As the amount of explosive used in practice is not proportional to the depth of the drill hole or the depth of undercutting, it would appear that the production of coal per unit of explosive used should be slightly increased for the deeper undercutting. Undercutting 6 to 7 feet, with adequate "snubbing," has in most mines been found satisfactory. The undercutting is always done in the coal.

The saving of explosive by proper undercutting is too often overlooked, although the advantage gained by giving the explosive the best possible chance to work can clearly be measured by the results obtained. If the coal is not "snubbed," it may rest rather firmly on the bottom, so that extra work is required to release it; and if undercutting is incomplete and small solid blocks, coal sprags, or cuttings are not removed from underneath, the shot can not properly do its work and a part of the advantage of undercutting is lost. The explosive force follows the lines of least resistance and can do only a certain amount of work.

Some machine men do not finish the undercutting square with the rib when using the short-wall machine. The machine is pointed into the rib at the back of the cut and then withdrawn at an angle. This prevents placing the rib hole to advantage, and as a result the left rib will break in a "zigzag" fashion. The miner in endeavoring to maintain a straight rib will angle the hole in toward the solid coal; extra explosive is required and the results are less satisfactory.

Data from two mines, shown in Table 3, give a proportionately larger production per unit of explosive with the 7-foot machine as compared with the 6-foot machine. In some instances less explosive was used after a 7-foot machine than after a 6-foot machine in rooms on adjoining entries. The miners noted no difference in the work of loading out the coal. It is thought that the 7-foot cut is the maximum for good results in this field.

The method of snubbing back half the distance of the undercut is a good one. At some mines hand snubbing is rather difficult but for the most part the coal is suitable for snubbing and is easily broken down in slabs. Often the bottom layers of coal immediately above the undercut will break down from the weight, thus assisting snubbing. Snubbing benefits both the miner and the operator. It is to the miner's advantage because it saves explosive and makes the work of obtaining the coal easier. Usually the face breaks at an angle, with the top projecting out 2 feet farther than the bottom. When snubbing is properly done, the coal when shot falls forward and rolls over, thus giving the miner the same advantage as in loading after puncher machines. As there is usually a differential in the rates for loading after chain machines and after puncher machines, and as coal mined with chain machines and snubbed presents practically the same conditions as coal undercut with puncher machines, the miner can consider that he is being paid for snubbing at the rate of the number of tons of coal loosened per round of holes times the difference between cost of loading per ton after puncher and after chain machines. In Franklin County this difference amounts to 6 cents per ton; the amount of coal shot down in a room 24 feet wide, 7½ feet high, and a 7-foot undercut averages about 50 tons; then 50 times \$0.06 equals \$3, which represents the value of snubbing. Snubbing also gives better sizes of coal, which is an advantage to all parties concerned. Frequently, however, the miner neglects to do enough snubbing. At some mines the shot firers refuse to shoot shots when the snubbing is inadequate; at other mines face bosses are employed who in addition to other duties supervise snubbing.

ADVANTAGE OF FIRING "SNUBBING SHOTS."

In a number of mines where the coal mined averages 8½ to 9 feet, and up to 10 feet, thick "snubbing shots" are fired. Inasmuch as the limit charge of permissible explosive is often exceeded under average conditions, snubbing shots are necessary in the thicker coal and could be employed to good advantage throughout the field. In the mines in Franklin County the average thickness of coal worked is about 8 feet. The top coal, in which holes are drilled under present methods, is a brittle coal. Sufficient explosive must be inserted to break down 7 to 8 feet of coal. The bottom bench below the "blue band" ranges from 1½ to 3 feet thick and is by far the hardest part of the seam; in some sections the "blue band" is 2 to 3 inches thick and in places very hard to break. Thus it can be seen that under present methods the firing of three shots to break this entire thickness of coal, with the hardest part of the seam the farthest from the explosive and with the explosive embedded in the more



A. FRESHLY SHOT FACE OF NO. 6 COAL IN FRANKLIN COUNTY MINE USING PERMISSIBLE EXPLOSIVES.



B. RESULT OF SHOT WITH PERMISSIBLE EXPLOSIVE IN FRANKLIN COUNTY MINE, SHOWING COAL ROOF.



C. LUMPS OF COAL PRODUCED WITH PERMISSIBLE EXPLOSIVES, NO. 6 BED, FRANKLIN COUNTY.



A. FACE OF ROOM IN NO. 6 COAL, FRANKLIN COUNTY MINE, AFTER "SNUBBING SHOTS" HAVE BEEN FIRED. COAL BROKEN AWAY AT THE "BLUE BAND,"



B. ROOF OF NO. 5 BED AFFECTED BY FIRE. A FREQUENT CONDITION IN MINES IN WHICH BLACK BLASTING POWDER IS USED.

brittle top coal, requires excessive charges and such charges must tend to increase the proportion of screenings.

Plate VIII, A, shows the results of snubbing shots in coal 9 feet thick. The coal underneath the "blue band" has been broken down, giving excellent lump. After this has been loaded out, or even partly loaded out, the remaining part can be shot down with good results. A special tool, an iron rod three-fourths of an inch in diameter and about 6 feet long, with a handle on one end and a prong or hook on the other, can be used efficiently for pulling out the coal produced by the snubbing shots. This total is known as a snubbing or coal hook.

MINING "ON THE BENCH."

In order to furnish for this report some data on the practicability of mining "on the bench" in the No. 6 bed, a test was conducted at the West mine, at West Frankfort, during the latter part of March, 1916, by Robert Forsyth, superintendent of the West Frankfort Coal Co., who first proposed trying out such a method as early as 1909, at the time explosions were so frequent in this field. The method, although not new for Illinois, had never been tried previously in Franklin County. The accompanying sketches in figure 16 show the features of this method.

This test was made in rooms 3 and 4, off the third southeast stub entry. In order to establish the "bench," the mining machine was unloaded onto a specially erected platform, so as to raise the cutter bar to the desired height. The first cutting was made across the face of the room immediately above the "blue band" in the soft coal, 5 to 6 inches thick, occurring just a few inches above the "blue band," as in A, figure 16. The coal was then snubbed by hand 1 foot high and 2½ feet deep, as is the regular custom, and shot down with three holes with two and one-fourth sticks of explosive in the center hole and two sticks in each rib hole. In regular shooting with three holes, when the mining is on the bottom, four to five sticks of explosive per hole are used. This fall was then loaded out and another cutting put in, as shown at B, figure 16. This time, however, the machine was unloaded on the bench, and the cutting was done in the usual way. The height of coal above the bench was 5 feet 5 inches. Another round of shots was fired, and as the bench had already been established, this coal was loaded from the bench directly into the car, the bench being 2 feet 4 inches high, thus giving the miner the advantage of much easier loading. After this coal was loaded out, a third cutting was made and three holes again drilled and charged, as shown at C, figure 16.

The plan then required shooting up one cut of bottom in order that the face of the coal could be kept at a uniform distance from the

track, so three holes were accordingly drilled in the bottom coal, and one stick of explosive was placed in the center hole and three-fourths of a stick in each rib hole. This shot the bottom very successfully, giving practically all lump coal, as the shots merely loosened the coal in large lumps. The bottom holes were placed as shown in C

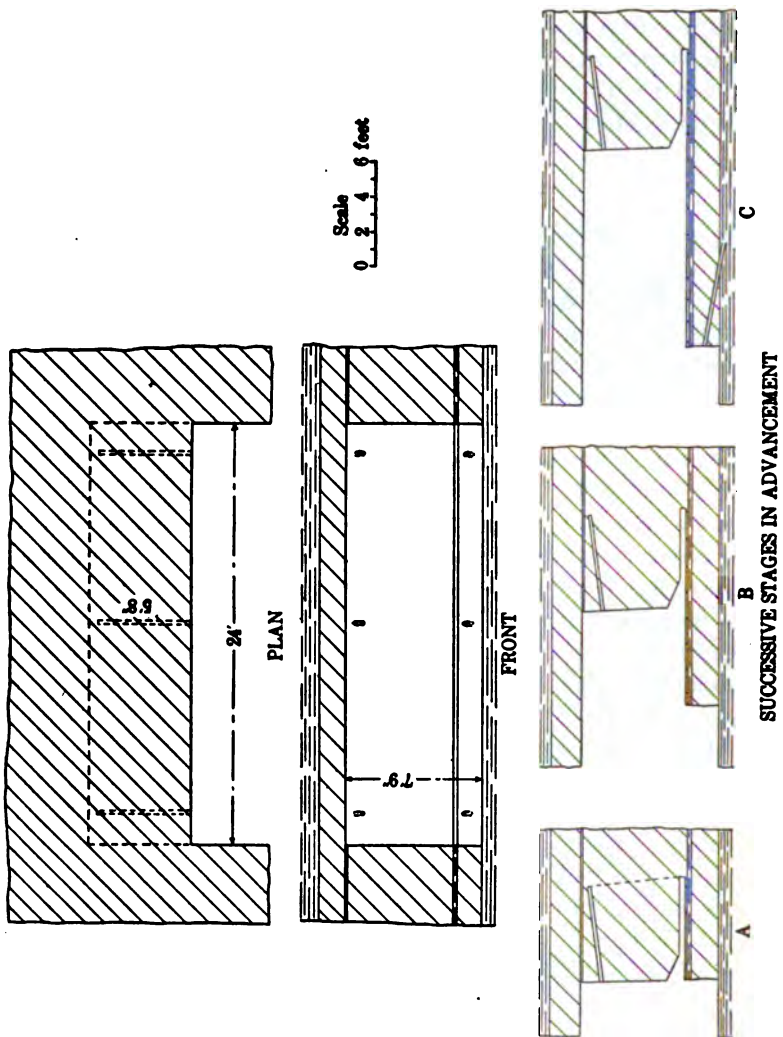


FIGURE 16.—Mining on the bench, showing successive stages. From tests in a Franklin County mine to determine results obtained from using permissible explosives.

and in the front view, sloping downward into the bottom clay from a starting point about 6 inches from the top of the bench. The center shot in the bottom coal in room 4 was drilled downward from the top of the bench, about 4 feet back from the edge, at an angle into the clay. Tabulated results of the shots follow:

Results of shots with a permissible explosive test of mining "on the bench" in a Franklin County mine.

Location.	Width of room.	Total thickness of coal.	Thickness of coal shot.	Depth of undercutting.	Saubbing.		Position of hole.	Depth of hole.	Distance of hole from roof.	Distance of hole from rib.	Number of sticks of explosive used for each shot.	Total sticks of explosive used.	Total pounds of explosive used.	Number of tons of coal produced. ^a
					Height.	Depth.								
Room 3, 3 southeast stub entry:	Ft.	Ft. in.	Ft. in.	Ft.	Ft.	Ft.	Left.....	Ft. in.	Ft. in.	Ft. in.				
Top shots ^b	24	7 9	5 5	5½	1	2½	Center.....	4 8	0 10	1 0	2			30
Bottom shots.....			2 4				Right.....	4 8	0 10		2½	6½		
Total.....							Left.....	4 0		1 0	1½	2½		11
Tons per 25-pound box of explosive.....							Center.....	4 0			½			41
Room 4, 3 southeast stub entry:							Right.....	4 0				8½	4.5	22½
Top shots.....	26	8 1	5 8	5½	1	2½	Left.....	4 9	1 0	1 3	2			32½
Bottom shots.....			2 5				Center.....	4 9	1 0		2½	6½		
Total.....							Right.....	4 9		1 4	2			13
Average tons per 25-pound box of explosive.....							Left.....	5 0		1 2	1½	3½		49½
							Center.....	3 6			1½	10	5.16	222
							Right.....	4 10		1 2				

^a Average tons produced per 25-pound box of explosive, 223.5; average for these rooms under the present method is about 17½ tons.^b Top holes drilled within 3 inches of roof.

It is felt that as so large a part of the future production of coal in Illinois will be from coal No. 6, any innovation in mining methods that will encourage the use of permissible explosives and lead to production of less fine coal and possibly increase the miner's earnings, besides giving other advantages, should be carefully considered.

The advantages of the system described might be enumerated as follows: Possibly larger sizes of coal can be produced, and with less explosive. Loading off the bench makes work easier for the miner, as the coal on the bench can be shoveled directly into the car and need not be raised the full height of the car, and the "blue band" can be removed more easily. A row of props can be carried on the bench close to the face, thus insuring greater safety and fewer accidents from roof falls; also a permanent row of props can be kept up to the bench, and the miner can stand on the bench and set these props to advantage. The miner can examine the roof more thoroughly; the undercutting can be done more quickly, as the coal is softer at this point; and bits will require only about one-third as much sharpening as when cutting in the hard bottom coal. Drilling will be easier for the miner standing on solid coal than, as at present, standing on a box while drilling coal $7\frac{1}{2}$ to 8 feet high. The use of smaller charges of explosive will be easier on the roof and break it less, an extremely important matter in Franklin County where the coal roof is left up; the limit charge of $1\frac{1}{2}$ pounds of explosive will not be exceeded, which overcomes the objectionable feature of placing heavy charges of explosive in top coal, as is necessary in breaking down $7\frac{1}{2}$ to 8 feet of coal with a hard bottom bench. No changes need be made in the types of mining machines now in use except that possibly wheels of larger diameter, 18 to 22 inches, might be employed for the machine truck, so that the mining machine can be unloaded directly onto the bench when the bench is once established. At times only the middle bottom shot need be fired with this method, and then the bench is not only in front of the car but along the sides as well, as space can be made for extending the track.

The principal disadvantages of this method are that, from the miner's standpoint, it will require the drilling of three holes more than by present methods and possibly better workmanship in taking up the bottom coal in some places. Also it will make a change in conditions. However, its advantages are believed to overbalance greatly any disadvantages. The use of electric drills would obviate such extra work, and these drills should be used, provided that suitable changes in the wage scale were made.

METHODS OF USING EXPLOSIVE.

It is generally conceded that it requires an experienced miner to get the best results from permissible explosives, as from black blasting powder; it is also the belief that a miner has advanced in his

vocation when he can also use permissible explosives successfully. The miner who has become accustomed when using black blasting powder to angle the holes into the ribs and drill them slightly on the solid finds that permissible explosive will not give the best results under such conditions. When permissible explosives are used, the hole should be practically parallel with the rib and have sufficient clearance from the rib and from the back of the undercutting. The inclination of the hole may vary slightly.

The excessive use of explosive increases both the cost per ton of coal and the amount of screenings produced. The tendency to use too much explosive is one of the most detrimental factors to the success of permissible explosives in any mine. The miner's inherent dislike to spend time picking down coal after a shot tends to result in shots being overcharged except by miners who consider the saving from using the proper amount and the danger to life and property from an overcharged shot.

The sizes of the sticks of explosives as used in Illinois mines are $1\frac{1}{2}$ by 6 or $1\frac{1}{2}$ by 7 inches. The limit charge, $1\frac{1}{2}$ pounds, would restrict the amount used to three sticks or less per shot, giving a charge 18 to 21 inches long, which would occupy about one-third of the drill hole. As noted in Table 2, the number of sticks in a 25-pound box of explosive varies from 40 to 53, depending upon the size of the cartridge and the density of the explosive. In a 25-pound box containing 40 sticks of explosive, the average weight per stick is 10 ounces; in a box containing 50 sticks, the average weight per stick is 8 ounces; therefore, it follows that two and one-half sticks of the former would do the same work as three sticks of the latter, assuming that the sticks of explosives are of equal strength. For this reason it is often to the miner's advantage to cut a stick into halves or even into quarters. The accurate judging of the amount of permissible explosive to use is highly important with respect to the sizes of coal produced.

Closely associated with the improper use of explosives is the dissatisfaction that may result from misfires, or failure of shots, which are due to a number of causes almost wholly within the province of the miner to prevent. Preventive measures include care in the storage and handling of explosive, fuse, and detonators, and the careful preparation of shots, as previously mentioned. Misfires are costly, resulting in smaller output, waste of explosive and loss of time and increase of danger in drilling and charging a new hole.

It has been suggested that it would be advisable for operators to provide for the supervision of snubbing and of the arrangement and charging of shots. Often the shot firers have assisted in this work, giving instruction where necessary, and have thus aided in bringing about better results, to the miners by improving methods of work

and to the operator by increasing the proportion of the larger sizes of coal, and in improving safety. The shot firer by virtue of his work is in a position to command the confidence of the men, to detect improper practices, and to assist in establishing safer practices. The careful selection of shot firers has had much to do with the successful use of permissible explosives.

ORDER OF FIRING SHOTS.

The system of lighting three shots at a time in a room, when firing with fuse and detonator, is somewhat general. This system tends to increase the proportion of screenings and also increases the amount of explosive necessary and consequently the explosives cost to the miner, for often the "buster" or center shot, which is given a shorter fuse and is fired first, will break the coal nearly to the adjoining rib shot, leaving only a relatively small amount of coal to be broken down by the previously charged rib shots. The rib holes are charged on the assumption that a certain amount of coal will be left standing, whereas if the miner could see the amount to be shot, he could gage the charge for the rib shots more correctly, and thus save explosive. Therefore there is an advantage in firing the "buster" shot before the rib holes are charged; in other words, firing the "buster" one night and the rib shots the following night, and this should be done at every opportunity. The coal from the "buster" shot need not all be loaded out before the rib shots are fired.

As each two miners have two working rooms, also a crosscut through the pillar every 60 feet, such practice can be frequently followed. Usually, however, in order to avoid delay in undercutting, the face in one place is squared up and made ready for the machine men before any coal from shots in the adjoining room is loaded out, and at times it could not be followed.

At one mine the rule is to charge and fire not more than two shots in a room on any one night. This has the advantage of supplying ample coal for a day or more and the charge required for the remaining rib shot the following night can be accurately judged. In the various mines a depth of undercutting of 6 feet on a face about 24 feet wide yields approximately 40 tons; a 7-foot cut yields 50 tons. Each place is cut and shot on an average not oftener than twice each week, which with two men and two rooms provides 160 to 200 tons to load, not including the coal from the intervening crosscuts. The average daily tonnage ^a per miner in these fields indicates that such a system can be followed, and often the firing of a single shot per night would be practicable. Table 3 gives data on the average production of coal per shot.

^a Data by Andros show a daily output of 7.6 tons per loader for the sixth mining district. See Andros, S. O., Coal mining in Illinois: Bull. 13, Illinois Coal-Mining Investigations, Cooperative Agreement, 1916, p. 148.

Data from a typical mine for the year 1915 follow:

Number of tons produced per day by each loader in an Illinois mine during 1915.

	Tons.
January.....	11. 48
February.....	11. 20
March.....	11. 99
April.....	12. 19
May.....	11. 50
June.....	11. 16
July.....	11. 63
August.....	(a)
September.....	11. 55
October.....	11. 93
November.....	10. 75
December.....	10. 90
Average.....	11. 48

WIDTH OF WORKINGS.

The working dimensions are fairly uniform in the mines listed, the rooms average 24 to 26 feet wide; the width of room necks varies. Including room necks and crosscuts, the usual ratio of wide places to narrow work is about three to one. In one mine coal was at one time being produced from 220 rooms and 68 entries; in another mine, from 186 rooms and 50 entries, not including entry and room crosscuts; in another, 170 rooms and 30 entries. In a number of the Franklin County mines where the rooms are driven their full length, the last crosscut is made 18 feet wide at the face, connecting all rooms and making a continuous face, which gives a larger proportion of wide work, and consequently helps to increase the percentage of lump coal.

In one new mine in Franklin County, data on all development work 12 feet wide shows an output of approximately 100 tons of coal per 25-pound box of explosive, the percentage of 2-inch screenings being slightly over 50 per cent. It is believed that when the mine is being operated at full capacity, the proportion of screenings will be decreased 7 to 10 per cent with a corresponding increase in the larger sizes. Data from one mine fully developed, in which 257 rooms averaging 24 feet wide and 70 entries averaging 12 feet wide, were being worked, show that 94 per cent of the coal is from rooms and room crosscuts, and 6 per cent from entry and entry crosscuts.

Data given in previous pages show that the cost of blasting per ton of coal in entries is practically double that of wide rooms. Table 4 shows that the average production per 25-pound box of explosive is 209 tons in rooms and 104 tons in entries, and that in the various typical mines the production per unit of explosive increases with the width of workings.

^a No data available.

Coal produced from pillars or from areas affected by squeezes yields a high percentage of screenings, but in the mines under discussion the amount of such coal is practically negligible as few pillars are drawn and little coal is obtained from squeezed areas. The amount of top coal recovered is also very limited.

The relative proportions of different sizes of coal, which are affected to more or less extent by these various factors, are also influenced by breakage resulting from the handling underground, and in the various methods of preparation. Holbrook's report ^a on the preparation of coal at Illinois mines contains interesting data on this subject.

In the foregoing, recognition of the many obstacles that the miner and management meet and the difficulty in overcoming them have not been lost sight of, but the fact remains that some of these are inherent to mining. Each division of underground work bears a relation to every other division, and all should move in harmony toward the objectives—greater safety, less explosive per ton of coal produced, and larger sizes of coal.

ADVANTAGES OF PERMISSIBLE EXPLOSIVES.

Rutledge and Hall^b mention the advantages of permissible explosives over black blasting powder, as found in practice throughout the various mining districts as—lessened risk of fire, making fire runners unnecessary, less damage to roof, cleaner ribs, fewer blown-out shots, less scattering of coal, fewer air blasts (shocks to the mine air), and windy shots, permissible explosives better adapted for use in wet holes, absence of smoke, pleasanter and safer mining.

These advantages have been clearly shown in Illinois mining practice and are discussed relative to the following: Fires and explosions, blown-out and windy shots; effects on roof, accidents, effect on general mine efficiency.

MINE FIRES AND EXPLOSIONS.

The occurrence of fires and explosions from the use of black blasting powder has been pointed out in preceding pages. No data have been kept of costs of the various fires, but even should no widespread disaster occur through them, yet the frequent occurrence even of small fires is a source of great anxiety and expense. When a fire is discovered in its early stages it is easily smothered, but often it may smolder in newly shot coal and the fire runner may pass by without detecting it. If the fire is under headway it is often necessary to seal off a considerable area of the mine, which results in loss of wages

^a Holbrook, E. A., The dry preparation of bituminous coal at Illinois mines: Bull. 88, Eng. Exp. Station, University of Illinois, 1915, p. 133.

^b Rutledge, J. J., and Hall, Clarence, The use of permissible explosives: Bull. 10, Bureau of Mines, 1912, pp. 20-25.

and output, and in addition to this entails the cost of sealing, opening, and cleaning up several hundred feet of entry, and the dangers attending this work. Especially in mines having a high volatile coal, bad roof, and generating explosive gases, the problems created are expensive and serious. The subsequent bad roof conditions are an additional source of expense and danger. Plate VIII, *B*, shows the after effects on the roof of a fire from a black powder shot in a mine working No. 5 coal. Practically all of the mines in which permissible explosives are now used and which formerly used black blasting powder have had trouble and expense from fires. At one mine 20 fires occurred, each of which cost \$100 to \$500 to combat; at another mine 12 to 15 fires occurred, which stopped mining one to three days and cost \$50 to \$300 each to fight. These figures do not include losses in wages and output. At another mine several large fires have occurred. On one occasion both shafts had to be sealed for a period of nearly seven weeks. Within two months another fire started which required the erecting of five stoppings and the sealing off of 80 working places, with all miners' tools and one mining machine. The cost of sealing off this fire was \$1,100, to which must be added the subsequent and indirect costs. A tabulation of the costs of fires in the various mines would form an item possibly aggregating, in proportion to the value of the property involved, the cost of fires in some of our larger cities.

The necessity of providing fire runners in the various sections where black blasting powder was used created a daily expense item of considerable amount.

In so far as the elimination of fires and explosions is concerned, it can be stated that the use of permissible explosives has effectively met the situation in the districts where used.

It is true that small fires have sometimes resulted from the use of permissible explosives, but these have been caused chiefly by the improper use of such explosives, from a number of causes, as follows: (1) Using more than the limit charge of $1\frac{1}{2}$ pounds of explosive (any explosive when used in excess becomes more dangerous); (2) using insufficient stemming or no stemming, especially in the presence of gas feeders; (3) the use of damp explosives or weak detonators, either of which may cause the explosive to burn instead of detonate or may cause a detonation of a low order which would destroy the main feature that makes an explosive permissible; (4) the use of explosive which has changed in composition through age or in which the nitroglycerin has segregated through long or improper storage so as practically to change its composition; (5) the use of two kinds of explosive in one drill hole; and (6) the use of damaged fuse, the "side spitting" of which may ignite a gas feeder or else ignite the explosive, causing an initial burning.

In nearly every instance in which fires from explosive have been reported in Illinois mines using permissible explosives, evidence was at hand to point to one of the above causes. At one mine a small fire followed a "tight" shot on the rib. The miner had kept his box of detonators buried under several inches of dirt and the detonators had become moist. The charge was undoubtedly incompletely detonated and burned in the drill hole. At another mine the use of old explosive caused a fire; at another three fires from one of the permissible explosives first placed on the market occurred during the first few months that permissible explosive was used, but the cause is not known. In two other instances, fires were reported, due in one instance, it was thought, to the fuse "side spitting" and in the other to the explosive burning. Only few fires were reported during the past year in the Illinois mines using permissible explosives, although approximately 800,000 shots were fired. No explosions have occurred in several millions of shots.

The importance of care in the use and storage of explosives applies directly to the prevention of fires. In one mine in which during 1916 two fires occurred from shots of permissible explosive an investigation was made and it was found that in the storage of explosives in the magazine a fresh shipment had been stored on top of much older explosives, resulting in some of the explosives being much too old for safety or efficiency when finally used. When a new consignment of explosives is received, it should be stored in the magazine in such a way that the oldest explosives can be issued first. An instance is reported of a carload of explosive being kept on hand for over a year and only four boxes were discarded, the remainder being used after this time; this, however, is an exception and it is generally believed that explosive over six months in age should not be used. If a carload can not be used in this time, the explosive should be purchased in smaller quantities.

Some of the causes of fires as given above refer to the use of old or deteriorated explosives—explosives that would fail to pass the tests for permissibility. In others it is assumed that the explosive used is in all respects similar to the sample that passed the required tests at the Pittsburgh testing station of the Bureau of Mines—that is, it is a standardized permissible explosive. During the winter of 1916-17, at one mine there was trouble from a certain explosive that burned in the drill hole. A number of small fires resulted and one fire which got beyond control had to be sealed. As the cause was not apparent and the fires were confined to a particular mine, the manufacturer was asked to investigate the matter. Later the trouble was overcome. This experience again demonstrates the readiness with which the coal face in the mines in southern Illinois ignites and the desirability of using short-flame explosives only in

these mines; also, that in combating the fire hazard, the cooperation of all parties concerned is necessary.

Two field samples of the explosives sent from Illinois mines to the Pittsburgh testing station at various times exceeded the allowable tolerance, and the particular lots of explosives represented by such samples were therefore declared nonpermissible.

BLOWN-OUT AND WINDY SHOTS.

The characteristic action of permissible explosives helps to prevent blown-out and windy shots such as occur with black blasting powder. The explosive expends its energy on the coal and tends to crater or enlarge the hole rather than to blow out of the hole. Overcharges will not give windy shots with accompanying blowing out of props and brattices and derangement of ventilation, as would a corresponding overcharge of black blasting powder. The feeling of relative safety among shot firers in mines using permissible explosives sometimes tends toward a disregard of necessary precautions. Care should be exercised, and the explosive should always be in good condition, properly tamped, and fired with full strength detonators.

EFFECTS ON ROOF.

In comparing black blasting powder and permissible explosives as to their relative effects on the stratum that forms the immediate roof, some difference of opinion may exist. All mining men agree that any explosive may injure the roof. In Franklin County from 10 to 40 inches, usually about 24 inches, of top coal is left for roof to hold the shale which falls when exposed. The poor roof conditions typical of the No. 6 coal after the overlying shale becomes broken are illustrated in Plate I (p. 6). The effect of the explosive on the roof is therefore of considerable importance, as often bad roof conditions are a serious hindrance to mining.

Black blasting powder, like other explosives, exerts force in every direction, but as it acts slowly its gases can work upward through cracks and crevices. As the natural roof is harder than the coal, this effect is not readily observed. In some of the Franklin County mines soft friable shale containing many parting or bedding planes and crevices overlies the coal. The slowly generated gases from black blasting powder work up into these planes or if, in the case of "frozen" or sticking coal it is necessary to place the shot close to the roof, the powder gases "burn" through these crevices. When this occurs, in many cases the roof may apparently be in good condition after the shot and may stand for two or three months, and then break down from the combined action of air and moisture in the cracks enlarged and weakened by the shot of black powder.

The gases generated from explosives of the permissible types do not "work ahead" like black blasting powder, also do not work up into and weaken the roof. The gases are generated so suddenly and the energy is so quickly dissipated that the coal breaks much more quickly and the liberated gases can not work into these crevices, make new crevices, or enlarge the old ones. This feature of these explosives assists in making them "permissible."

In the No. 6 seam there are a number of partings in the friable top coal, and often when black blasting powder was used the gases did not follow the regular parting planes but worked far up into the top coal. This condition can be traced in a number of mines in the Franklin County field, where black blasting powder was formerly in use; in fact, the places along the entries can be noted where the change from black blasting powder to permissible explosives was made. The flame of black blasting powder would spread out from the end of the drill hole and follow along the different partings, especially when the hole sloped upward too much. The characteristic action of permissible explosives is noted when the hole projects upward into the plane beyond that intended; in this event "holing out" results—which, however, is of limited extent. Miners often drill upward, or into the ribs, more than is desirable for the purpose of gaining a greater production per unit of explosive, and this tendency has to be guarded against on account of the greater amount of timber necessary. The character of the roof of the No. 6 coal and a method of timbering an entry are shown in Plate I (p. 6), which illustrates a problem met in all these mines.

Experience in Franklin County and other mining districts shows that permissible explosives are much easier on the roof than black blasting powder. In a Pennsylvania mine, in a powder test, a series of about 100 shots was fired with black blasting powder, and then a series with a permissible explosive. There was a "draw slate" about 9 inches thick above the coal, which was mined with the coal, and the holes were drilled so that the point of the hole came close to the parting between the coal and this "slate." The conditions as to the placing of holes were observed for each shot, and all holes were charged under the supervision of a committee of miners and operators. It was found that in almost all cases black blasting powder caused the "slate" to fall with the coal, or shortly thereafter, whereas when the coal was shot with the permissible explosive the "slate" invariably had to be shot separately. This extra blasting may have increased the cost slightly in this particular mine, but it shows that the use of permissible explosives undoubtedly tends to be easier on the roof, and thus lessen the danger of roof falls.

In the majority of mines where permissible explosives are used it is the opinion that they are much easier on the roof. In one mine

where poor roof is encountered, permissible explosives are giving much better results than were previously obtained from black blasting powder. In another mine the only instances in which results were unsatisfactory were in some entries where the dip of the bed changed and the entrymen drilled up into the roof before they were aware of the situation, resulting in a poor roof for some distance. At another mine, where the parting planes are not always pronounced, holes are often drilled across these planes, but the roof is not seriously broken; a small cavity may result, but not enough to require extra timbering. At still another mine, where permissible explosives are used, it was thought at first that the explosive was damaging the roof, but further observation showed that this was due to local conditions; and further on, where conditions became normal, these effects disappeared. Exceptionally good roof conditions were noted throughout one mine where permissible explosives have been used exclusively. In this mine the props were recovered from practically all of the rooms of one pair of panels that had been driven up to the boundary, and the roof held well without a break for some time. In some places the roof of a whole room, with an area of 24 by 250 feet, held for weeks after all the timbering had been removed. This is rather unusual for the roof of No. 6 coal. The superintendent at this mine estimates that the use of permissible explosives has saved his company many thousands of dollars in timbering.

In the No. 5 seam no top coal is left, and the coal is shot free from the roof, as has been previously stated. In one particular mine the roof is of light-colored shale, which gives a free parting and consequently a higher production of coal per unit of explosive. At three other mines no damage to the roof from permissible explosives has been noted. In only one mine of the entire district was it believed that permissible explosives had a more damaging effect upon the roof than black blasting powder, and it was thought that the miners could overcome this by properly placing and loading the holes.

FEWER ACCIDENTS FROM PERMISSIBLE EXPLOSIVES.

Permissible explosives are not easily ignited by a spark from a lamp, a frictional spark, or a flame from burning gas, and are not exploded by an electric current unless detonators are present, and hence there is less possibility of accident than when black blasting powder is used. Illinois mining history has recorded numerous instances of fatal and nonfatal accidents from the use, misuse, and transportation of black blasting powder, but in spite of the fact that only a small percentage of the miners were at all familiar with permissible explosives at the time of their introduction, the accident record is singularly free from accidents due to the use or handling of permissible explosives. Franklin County records show only two fatal accidents charged to

permissible explosives, one of which is said to have resulted from a miner prying with the point of his lamp hook into a detonator that had misfired. His act is supposed to have exploded the fulminate in the detonator, which in turn exploded some near-by detonators and a 25-pound case of a permissible explosive.

Defective squibs, powder ignited by burning gas, by a spark from a lamp, spark from smoking tobacco, by a spark in drilling out a missed shot, by a spark from a pick in opening a powder keg, or by an electric current or arc, all took their toll in lives. With permissible explosives practically all these dangers are removed, provided reasonable care is exercised in handling the sensitive detonator.

It has been frequently noted that the heat of an explosion of gas or dust readily sets off black blasting powder, which may increase the force of an explosion and at times renews one that has practically died out. During the fiscal year ended June 30, 1915, two explosions in Illinois mines were reported to have been started by the ignition of bodies of gas, black blasting powder being used in one of these mines and permissible explosives in the other. These accidents afford a comparison of the effects on the different explosives when exposed to the heat and violence of an explosion. In one of them numerous kegs of black blasting powder were exploded by the heat. These had been delivered the night previous and were full, unopened kegs, as shown by the unopened bungs, ignition having been caused by external heat. In the other, several boxes of permissible explosives had been thrown against the ribs and broken open and the contents scattered along the entry without the explosive being ignited or detonated. Evidences of extreme heat after both explosions were the remains of rolls of charred fuse, burnt clothing, paper, and coked dust. Furthermore, should any permissible explosive be set off under such conditions, on account of the factor of safety which its relatively short flame provides, there is much less likelihood of its contributing to the loss of life and property than of such loss being caused by black blasting powder. Detonators, however, are very sensitive to blows and they should be stored at a safe distance from the explosive. The fewer kept in the mine by the miner the better.

In the mines using permissible explosives, shots that have misfired are occasionally drilled out. No accidents have as yet been reported, but such practice is extremely dangerous. The best practice is to tamp the "missed" hole to its mouth and to drill and load a new hole at least a foot from the old one. In one instance noted the miner drilled a new hole because of a misfire, and after shooting the new hole found that the explosive from the first hole had not been detonated.

Additional evidence of the dangers of using black blasting powder was furnished by an accident in August, 1915, in a mine in southern

Illinois in which the miners tamped their own shots, and firing was done by shot firers. One miner, contrary to rules, approached the drill hole with his carbide light on his head, and as he pushed the cartridge of black blasting powder into the hole a quantity of gas was forced out and ignited by flame from his carbide lamp. As the miner ran down the entry the powder ignited and, exploding in the drill hole, gave rise to a "windy" shot which might have resulted disastrously.

EFFECT ON GENERAL MINE EFFICIENCY.

Under general mine efficiency may be included other advantages tending to make the use of permissible explosives satisfactory in these fields.

Fire runners are now unnecessary in these mines, whereas they were previously required when black blasting powder was used.

Permissible explosives withstand moisture better than black blasting powder and the explosives of the nitroglycerin class, especially, can be used in wet holes if the detonator is properly arranged.

There is less work in making up the charge with a permissible explosive and less danger. Cartridges are uniform in size and weight and the charge can be accurately gaged. This is important in undercut coal where equal charges of explosive are used daily.

A given quantity of a permissible explosive can do more work than a similar quantity of black blasting powder; hence fewer holes need be drilled and charged to shoot down the same amount of coal.

The quantities of smoke and noxious gases given off by permissible explosives are relatively small as compared with black blasting powder, and this, with the smaller quantity of explosive used per shot, is an immense advantage as regards the effect on the miner's health. No permissible explosive produces more than 158 liters (5.5 cubic feet) of poisonous gases per $1\frac{1}{2}$ pounds of explosive and most of them produce less than 120 liters (4.1 cubic feet). This excellent feature is overlooked to a great extent because all of the men except the shot firers are out of the mine at the time of firing, but is of additional advantage to the shot firers, as they can travel with greater safety. Shot firers are protected in their work and find the explosion hazard greatly decreased.

SUMMARY AND CONCLUSIONS.

The advantages of permissible explosives emphasize the important bearing that these explosives have on the prevention of accidents and property loss. Their introduction into Illinois mines was difficult, but, after a thorough trial, they have been accepted as the explosives most suitable for conditions in the respective mines in which they are now used. In making a change from one mine to another or from one grade of powder to another the miner must

necessarily familiarize himself with the new conditions in order to do efficient work. This applies directly to a change from black blasting powder to permissible explosives. With a better knowledge of the nature and merits of the permissible explosives, their users give many testimonials as to the advantage gained, a condition most pleasing to the advocates of greater safety. The consensus of opinion among mining men in these districts is that where the coal is undercut and snubbed, permissible explosives can be successfully used, with little, if any, increase in the production of slack.

The cost of explosive per ton of coal produced in these mines has been lowered, owing to the fact that less explosive is required to perform the same work than when black blasting powder was used, the sizes of coal compare favorably, and miners accustomed to the use of permissible explosives get as good results as they did with black blasting powder.

Results are more favorable than was anticipated by the early users of permissible explosives. It was thought there would be a large difference in the sizes of coal produced and in the cost of explosives per ton of coal, as compared with black powder. In mines where the value of the coal produced has been affected by an increase of 2 to 5 per cent in the production of screenings, it is felt that this is compensated by the increased safety through the elimination of fires and explosions, saving in wages of fire runners, fewer accidents in handling explosives, and the feeling of security which the use of permissible explosives gives; these considerations also offset the loss of the profit the operator previously gained by handling black blasting powder. In other mines no material loss has been sustained, whereas in certain mines it is believed that the less injurious effect of permissible explosives upon the roof more than balances any slight difference in sizes of coal produced.

The success of permissible explosives in Illinois mines shows that they have satisfied one of the great needs of the coal-mining industry. Their increased use throughout the United States has overcome many of the dangers which attended the use of black blasting powder. Their use ought to be extended so that the occurrence of mine fires, explosions, and accidents due to explosives will be reduced to a minimum. Permissible explosives stand for protection to life and property. Their use is no longer an experiment, but forms the basis for an effective and continuous campaign in accident prevention.

APPENDIX.

JOINT POWDER COMMISSION.

PROCEDURE IN TESTS WHEN MAKING CHANGE OF POWDER.

For the settlement of all powder questions that arise a joint powder commission is provided in the agreement between the operators and the miners. The procedure in all tests where a change of powder is desired is stated in the eighth clause of the agreement, as follows:^a

The price for powder per keg shall be \$1.75, the same to be delivered at the face when so requested. The miners shall purchase their powder from the operators, provided it is furnished of standard grade and quality; and provided further, that other permissible explosives may be used when necessary for the safety of life and property. The permissible explosives shall be furnished at the same relative cost as black powder, the interest of operator and miner alike considered. In determining the powder for a mine, it shall be the purpose to select powder which shall give the best results to the miner and operator alike. A majority of the miners on the pay roll at any mine may file with the operator a written complaint against the powder being furnished by the operator. Such complaint must explicitly state the objections to the powder and must be signed by a majority of the miners. The operator may also file with the pit committee written notice of his desire to change the powder being furnished, stating his reasons therefor. Upon the filing of either complaint or notice as above stated, a test shall be arranged for as soon as possible, and such test shall be conducted under written regulations hereinafter provided for or the regulations may be prescribed by the joint powder commission hereinafter provided for. A joint State powder commission, who shall serve during the life of this agreement, shall be constituted, consisting of six members, three of whom shall be appointed by each executive board, respectively, of whom not less than one on each side, shall possess technical knowledge and all of whom shall possess either technical or practical knowledge of the use of powder in producing coal. Said commission shall have full control of all powder tests not settled locally, as herein provided. The commission shall be empowered to investigate the various phases of the powder question and may negotiate with powder companies for the establishment of standard sizes of powder and the proper branding thereof. It shall be the duty of the commission to eliminate entirely from cases of powder disputes the participation, openly or otherwise, of representatives of powder companies and shall by all lawful and proper means seek to secure for both operators and miners uniform deliveries of such powder as is selected as the most suitable in any given case.

In case of a powder dispute arising at any mine as to kind of powder to be furnished under the provisions of this agreement, the dispute shall be first taken up locally by the interested parties and a local test made in order to determine whether the

^a Agreement by and between the Illinois Coal Operators' Association, the Coal Operators' Association of the Fifth and Ninth Districts of Illinois, and the Central Illinois Coal Operators' Association and the United Mine Workers of America, District No. 12, clause 8; continued in force April 1, 1916.

powder used or desired by either operator or miners is best suited or adapted for the use of the mine where the dispute arises. The local test shall be made in accordance with the following rules and regulations as nearly as possible and shall cover a sufficient time and tonnage to give the different powders tested a fair chance to determine their respective value: The powder to be tested shall be agreed upon by both parties to the controversy as being in the judgment of each best suited for use in the mine where the dispute arises, the interests of operator and miner alike considered. The kind of powder giving the best results in the opinion of the parties making the test shall be the powder used in the mine during the life of the contract unless otherwise directed by law or unless a radical change occurs in the character or condition of the coal seam or that a change of powder is mutually agreeable to both parties.

In the conducting of any local tests the following rules shall be observed and enforced, to wit:

1. Six members shall constitute the committee or commission for making any local tests, unless otherwise specifically agreed to by both operator and the miners; said committee or commission to be composed of three miners selected and authorized to represent the local and three men to likewise represent the operator. In case of additional members participating in any local test each side shall have equal representation.

2. The drilling of all holes shall be under the direct supervision of one miner and one operator, member of the committee or commission, during the conducting of any test.

3. One miner and one operator, members of this committee or commission, shall prepare all cartridges and keep a record of the measurements and weights of same, the results of the shots to be figured by weights of the powder used and the condition of the coal after being shot as a basis.

4. In no case shall the members of the committee or commission who drill or supervise the drilling of the holes have any knowledge of the powder to be used, or that has been used until after the shots have been exploded.

5. The members of the committee or commission shall tamp and fire or have direct supervision of the tamping and firing of all shots during any test.

6. A record shall be kept, giving the measurements of the face, the number of shots fired, the amount and the kind of powder used, together with the comparative condition of the place and shots.

7. The holes drilled shall be measured and inspected, when possible, by all members of the committee or commission and shall be placed as nearly as possible in the same direction and depth in order to insure as nearly as possible similar conditions and chances for the different grains or brands of powder to be tested.

8. In judging the results obtained from any test the interests of the miner and operator alike shall be considered, and if required by either party, a record of the tons of coal produced and the quality of same shall be kept.

9. The committee or commission shall have the right to require the sealing of any or all full or part kegs of powder in the mines during the conducting of any test and may adopt any other rule or method to insure that no other powder than the powder used by the committee or commission will count or figure in the results obtained by any test made by them.

10. The committee or commission shall eliminate entirely the participation, openly or otherwise, of men directly or indirectly representatives of any powder or powder company. In the event of a powder dispute arising, and the failure of either side to appoint its members of the committee or commission herein provided for within seven days after the complaint is filed, the State president of the organization failing to appoint its members locally shall make the appointment thereof promptly and the committee or commission so appointed shall proceed with the test as herein provided.

In case the committee or commission fails to agree after a test has been made locally, as herein provided, the said joint powder commission shall assume jurisdiction, and the whole data and records of the test shall be submitted to and be reviewed by it, and if all the conditions as above have been observed in conducting the test it shall take the matter up for final decision with power to make any further test it may deem necessary. It is understood and agreed that all the requirements of a local test as above must be complied with before the case can properly come before the joint powder commission.

Said commission may investigate shot firing devices, the merits of explosives other than powder, and the use of other mechanical agents than explosives for the production of coal, and make recommendations to the two organizations concerning the same.

It is understood that the question of package or kegs in which the powder is shipped be considered in the same way as the powder itself. That the joint powder commission will have access to the powder magazines at all times where disputes arise. That no new shipments of powder be stored on top of powder on hand, it being the intent to use the powder in the order in which it is received at the mine.

NOTE.—It is understood that the operator will not discriminate against any powder desired by the miners locally which meets the test as to grade and quality herein prescribed, provided it can be furnished at a competitive price and in sufficient quantities for the successful operation of the mine.

WORKING AGREEMENT BETWEEN OPERATORS AND MINERS.^a

That section of the working agreement between the coal-mine operators and miners of Franklin County relative to the use of explosives reads as follows:

(a) Whether the coal is shot after being undercut or sheared by pick or machine, or shot without undercutting or shearing, the miners must drill and blast the coal in accordance with the State mining law of Illinois, in order to protect the roof and timbers and in the interest of general safety. Any miner who clearly violates the letter or spirit of this paragraph may be discharged.

(b) The system of paying for coal before screening was intended to obviate the many contentions incident to the use of screens, and was not intended to encourage unworkmanlike methods of mining and blasting coal, or to decrease the proportion of screened lump, and the operators are hereby guaranteed the hearty support and co-operation of the United Mine Workers of America in disciplining any miner who from ignorance or carelessness, or other cause, fails to properly mine, shoot, and load his coal.

(c) That all "bug dust" or machine coal cuttings when practically free from impurities be loaded out with the "snubbings" or other coal so as to produce a merchantable mine-run coal.

The above does not contemplate any change in the present method of handling "bug dust" or machine cuttings in Franklin County, or other mines where it is necessary to load same out before shooting the coal, as a protection against explosions or fire.

Where the operator desires the "bug dust" loaded out separately this shall be done by the miner working in the place during his regular shift at the regular tonnage price and the company shall furnish cars promptly to load same.

(d) Wherever it is practicable the miners shall shoot the coal with two pounds of powder or less. Where a dispute arises as to the practicability of shooting the coal

^a Working agreement by and between the Illinois Coal Operators Association, the Coal Operators Association of the Fifth and Ninth Districts of Illinois, and the Central Illinois Coal Operators Association, and the United Mine Workers of America, District No. 12, sec. 5.

with two pounds of powder, such dispute shall be taken up and settled by the two organizations as provided in this contract.

(e) It is hereby agreed that the cost of firing shots during the life of this contract shall not exceed the cost per ton for the same work during the previous contract. It is understood that this clause does not mean that the operator can avoid paying for the work actually necessary to be done by the shot firers.

It is also agreed that there shall be no shot firers in mines where coal is undercut by hand or machines, except as mutually agreed. Where conditions in the past have necessitated shot firers they will be continued; where conditions develop that they are not necessary, they can be discontinued.

(f) The shot firers shall go into the mine two hours before the regular quitting time to satisfy themselves by examination and inspection of the shots to be fired that they have been properly placed and prepared. When examining shots the shot firers shall work single; when firing shots they shall work double.

(i) The miners shall continue to assume all responsibility heretofore resting upon them for the care of the working places and the proper character and placing of the blasting shots; and that in case of wet holes that will not stand if charged and tamped by the miners, the shot firers will attend to charging and tamping said holes.

STATE LAW IN REGARD TO SHOT FIRERS.

The Illinois law in regard to shot firers (Coal Mining Laws of Illinois, sec. 19) reads as follows:

SHOT FIRERS IN COAL MINES.

In all mines in this State where coal is blasted, and where more than two pounds of powder is used for any one blast; and, also, in all mines in this State where gas is generated in dangerous quantities, a sufficient number of practical, experienced miners, to be designated as shot firers, shall be employed by the company, and at its expense, whose duty it shall be to inspect and do all the firing of all blasts, prepared in a practical workmanlike manner, in said mine or mines.

The shot firers shall, immediately after the completion of their work, post a notice in a conspicuous place at the mine, in which shall be indicated the number of shots fired; also the number of shots they did not fire, if any, specifying the number of the room and designation of the entry, and giving reasons for not firing same. In addition they shall also keep a daily permanent record, in which shall be entered the number of shots or blasts fired, the number of shots or blasts failing to explode, and the number of shots or blasts that in their judgment were not properly prepared and which they refuse to fire, giving reasons for same; the record to be in the custody of the mine managers and to be available for inspection at all times by parties interested.

The superintendent or mine manager shall not permit the shot firers to do any blasting, exploding of shots, or do any firing whatever until each and every miner and employee is out of the mine except the shot firers, mine superintendent, mine manager, and man or men necessarily engaged in charge of the pumps and stables: *Provided, however*, that nothing in this section shall be construed to prohibit the employment in such mine of a reasonably necessary number of men during such time for the purpose of securing the workings in case of fire therein.

No miner or other person shall alter or change any drill hole, by increasing its depth, diameter or otherwise, after the same shall have been approved by the shot firer.

No shot firer, whether voluntarily, or by command or request of any person, shall fire any unlawful shot, or any shot which in his judgment, exercised as aforesaid, from his inspection thereof, made as aforesaid, shall not be a workmanlike, proper, and practical shot.

On firing ahead of the air the law (sec. 14, *d*) reads as follows:

Away from the pillar for the mine bottom, crosscuts between entries shall be made not more than 60 feet apart without permission of the State inspector of the district and then only in case of "faults." When such consent is given, brattice or other means must be provided within 60 feet of the face to convey the air to the working place until a crosscut is opened up.

When undercut or sheared, the entry, crosscut and room neck may be advanced concurrently, but not more than one cutting shall be shot in the room neck until the crosscut is finished; and after the entry has advanced 15 feet beyond the location of the new crosscut, only one shot shall be fired in the entry to two in either or both the crosscut and room neck at the same shooting time.

When not undercut or sheared, the entry and crosscut may be advanced concurrently, but no room shall be opened in advance of the last open crosscut, and after the entry has advanced 15 feet beyond the location of a new crosscut only one shot shall be fired in the entry to two in the crosscut at the same shooting time.

Not more than three shots shall be exploded at one shooting time ahead of the last open crosscut.

TOLERANCES FOR PERMISSIBLE EXPLOSIVES.

At a conference on June 7, 1915, between manufacturers of explosives and engineers and chemists of the Federal Bureau of Mines an agreement was reached providing for reasonable limits of variation or "tolerances" in the results of analyses and tests of permissible explosives. If these tolerances are exceeded by a given lot of any permissible explosive it is not permissible. The tolerances established, which were made effective July 1, 1915, have been published in Bulletin 96 ^a of the bureau, and are as follows:

In order to define more exactly what is meant by the phrase "similar in all respects" in the definition of a permissible explosive, namely, "an explosive is called a permissible explosive when it is similar in all respects to the sample that passed certain tests by the United States Bureau of Mines, and when it is used in accordance with the conditions prescribed by this bureau," the following tolerances are recommended for field samples or manufacturers' samples of explosives, beyond which such lot of explosives can not vary and still be considered permissible for use in coal mines: *Provided*, That where the Bureau of Mines finds a sample which does not come up to the tolerance limits, the bureau shall simply declare that particular lot of explosives not permissible, and a copy of the notification to the consumer or owner shall be furnished the manufacturer, the notification to state that the explosive did not meet the tolerance requirements for moisture or ingredients, etc., as the case might be.

Chemical analysis.—Moisture, to be fixed by a sliding scale of from 1½ per cent at zero to 4 per cent at 10 per cent of moisture in original sample, this tolerance being on total percentage of moisture in the explosive.

Other ingredients (or their equivalents) in quantities not exceeding 60 per cent, according to curve (shown in fig. 17). For ingredients in quantities of 60 per cent or more, the tolerance shall be plus or minus 3 per cent: *Provided*, That the ingredients of a permissible explosive shall be considered to be those substances reported as found by the Bureau of Mines in the original sample of that explosive submitted for test as to its permissibility: *And provided further*, That an equivalent shall be considered to be a substance which would not materially alter the properties of the explosive and which would produce the same result as the original substance.

^a Storm, C. G., *The analysis of permissible explosives*: Bull. 96, Bureau of Mines, 1916, pp. 80-81.

Products of combustion (determined by Bichel gage tests).—The volume of poisonous gases from 680 grams of the explosive, including its wrapper, must be less than 158 liters, except that in case the first test yields 158 liters or more of poisonous gases per 680 grams of the explosive, including its wrapper, the average result of three tests agreeing within 5 per cent of each other shall be taken, and no explosive shall remain permissible when this average for poisonous gases exceeds the above standard limit.

Physical tests.^a—

Rate of detonation (the average of three trials with Mettengang's recorder), plus or minus 15 per cent.

Unit defective charge (the average of three trials with the ballistic pendulum), plus or minus 10 per cent.

Grams of wrapper per 100 grams of explosive, plus or minus 2.0 grams (average of four determinations): *Provided*, That the manufacturers shall submit samples of all different sizes of cartridges, to be considered as part of the original sample, the amount of wrapper to be determined for each size of sample: *And provided further*, That the

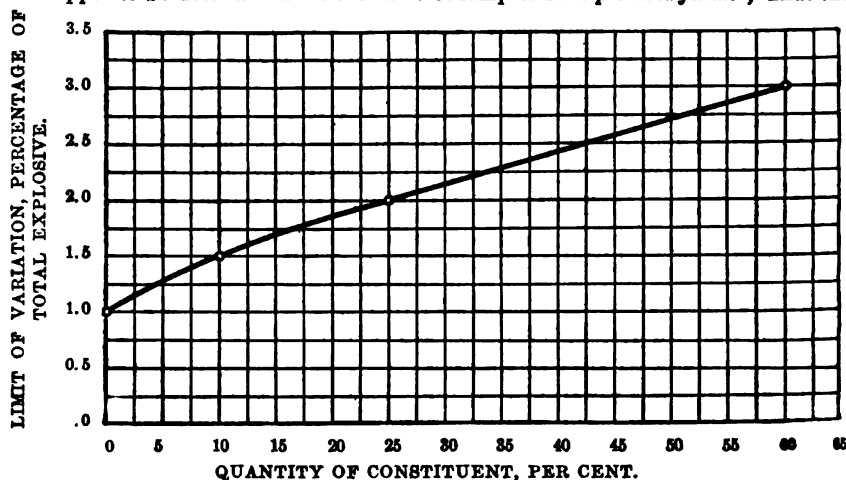


FIGURE 17.—Curve of limit variation in composition of permissible explosives.

tolerance as suggested shall be determined in comparison with the various diameters of samples as submitted with the original sample.

Apparent specific gravity of cartridge, by sand, plus or minus 7.5 per cent (average of four determinations): *Provided*, That actual density shall be determined on cartridges of the same diameter as the standard: *And provided further*, That manufacturers shall be required to submit samples of all sizes.

Gas and dust gallery No. 1.—No ignition must be obtained in each of one or more trials. Note: In the retesting of permissible explosives by tests 1, 3, and 4,^b the charges of the explosives fired will be reduced 10 per cent in weight from the weights originally used in order to eliminate any likelihood of a failure being due to the natural variations in the gallery conditions.

Pendulum friction test.—Each explosive must pass a test of 10 trials under the same conditions as originally tested, except that the height of fall of the wood-fiber shoe will be reduced by 10 per cent in order to eliminate any likelihood of a failure being due to the natural variations in test conditions.

^a For method of making these tests, see Hall, Clarence, and Howell, S. P., *Tests of permissible explosives*: Bull. 66, Bureau of Mines, 1913, 313 pp.

^b Hall, C., and Howell, S. P., Work quoted, p. 303.

BUREAU OF MINES TESTS OF PERMISSIBLE EXPLOSIVES.

The conditions under which manufacturers may submit explosives for test by the Bureau of Mines to determine their permissibility, the fees charged for the tests, and the test requirements of permissible explosives are set forth in Schedule 1 ^a of the bureau.

CONDITIONS UNDER WHICH TESTS WILL BE MADE.

The conditions under which the Bureau of Mines will test explosives to determine whether they shall be placed on its list of permissible explosives are as follows:

1. The manufacturer or applicant desiring tests to be made is to deliver to the Bureau of Mines, Fortieth and Butler Streets, Pittsburgh, Pa., three weeks prior to date set for tests, 100 pounds of each explosive that he desires to have tested. He is to be responsible for the care, handling, and delivery of this material to the testing station, and if he desires he may have a representative present during the tests. In order to avoid duplication of work, it is requested that the smallest size of cartridge that he intends to place on the market be sent for these tests.

2. No one is to be present at or participate in these tests except the necessary Government officers at the experiment station, their assistants, the representative of the manufacturer of the explosives, or the applicant desiring tests to be made.

3. These tests will be made in the order of the receipt of the applications for them, provided the necessary quantity of the explosive is delivered at the experiment station by the date set, of which date due notice will be given by the Bureau of Mines.

4. A list of the explosives that pass certain requirements satisfactorily will be furnished to the State mine inspectors in the several States and will be made public in such manner as may be considered desirable.

5. The details of results of tests are to be considered confidential and are not to be made public prior to official publication by the Bureau of Mines.

6. From time to time field samples of permissible explosives will be collected, and tests will be made of these explosives as they are supplied for use in coal mines in the various States. Field samples collected in original shipping case will be tested for their permissibility and will be analyzed; the proportion of each ingredient of the explosive must agree, within a reasonable variation, with that in the original sample submitted for tests.

TEST REQUIREMENTS OF PERMISSIBLE EXPLOSIVES.

The tests will be made by the engineers of the Bureau of Mines in gas and dust gallery No. 1 at Pittsburgh, Pa., or in one of the identical galleries. The charge of explosive to be fired in tests 1 and 3 shall be equal in defective power, as determined by the ballistic pendulum, to one-half pound (227 grams) of 40 per cent nitroglycerin dynamite in its original wrapper, of the following formula:

	Per cent.
Nitroglycerin.....	40
Nitrate of soda (sodium nitrate).....	44
Wood pulp.....	15
Carbonate of lime (calcium carbonate).....	1
	100

^a Fees for testing explosives and conditions and requirements under which explosives are tested: Schedule 1, Bureau of Mines, 1913, pp. 1-7.

Each charge shall be fired with an electric detonator (exploder or cap) strong enough to completely detonate or explode the charge, as recommended by the manufacturer. The explosive must be in such condition that the chemical and physical tests do not show any unfavorable results.

An explosive will be considered unsatisfactory if it is not chemically stable, if it shows leakage of nitroglycerin, or if it is in such condition that exudation of nitroglycerin would occur in handling or transportation.

A 680-gram (1½-pound) charge of explosive must not evolve 158 liters (5½ cubic feet) or more of poisonous gases as determined by gage tests. The poisonous gases that are likely to be evolved on explosion are carbon monoxide, hydrogen sulphide, and nitrogen oxides.

Explosives that are of such strength as to require more than 680 grams (1½ pounds) to satisfactorily break down coal under normal conditions of mining will be considered unsatisfactory. This is determined by the unit deflective charge (established with the ballistic pendulum) which shall not exceed 454 grams (1 pound).

An explosive will be considered unsatisfactory if in the course of the official tests two or more charges fail to detonate or explode completely, or if after two months' storage at the Pittsburgh experiment station two or more cartridges fail to detonate or explode completely when fired separately with a suitable detonator.

An explosive will be considered unsatisfactory if it is too sensitive to frictional impact (a glancing blow). This sensitiveness will be determined with the pendulum friction device with the shoe faced with fiber.

In order that the dust used in tests 3 and 4 may be of the same quality, it is always taken from the same mine, ground to the same fineness, and used while still fresh.

The following are the gallery tests to which are subjected the explosives that the Bureau of Mines is asked to place in the list of permissible explosives:

Test 1.—Ten shots each with the charge as described above, in its original wrapper, shall be fired, each tamped with 1 pound^a of clay stemming, at a gallery temperature of 77° F., into a mixture of gas and air containing 8 per cent of gas (methane and ethane). An explosive is considered to have passed the test if no one of the ten shots ignites this mixture.

Test 3.—Ten shots each with the charge as described above, in its original wrapper, shall be fired, each tamped with 1 pound^a of clay stemming, at a gallery temperature of 77° F., into 40 pounds of bituminous coal dust, 20 pounds of which is to be distributed uniformly on a wooden bench placed in front of the cannon and 20 pounds placed on side shelves in sections 4, 5, and 6. An explosive is considered to have passed the test if no one of the ten shots ignites this mixture.

Test 4.—Five shots each with a 1½-pound charge, in its original wrapper, shall be fired without stemming, at a gallery temperature of 77° F., into a mixture of gas and air containing 4 per cent of gas (methane and ethane) and 20 pounds of bituminous coal dust, 18 pounds of which is to be placed on shelves along the sides of the first 20 feet of the gallery and 2 pounds to be so placed that it will be stirred up by an air current in such manner that all or part of it will be suspended in the first division of the gallery. An explosive is considered to have passed the test if no one of the five shots ignites this mixture.

EXPLOSIVE NOT TO BE CONSIDERED PERMANENTLY PERMISSIBLE.

It must not be supposed that an explosive that has once passed the required test and has been published in lists of permissible explosives is always thereafter to be considered a permissible explosive, regardless of its condition or the way in which it is used. Thus, for example, an explosive named in the permissible list, if kept in a

^a Two pounds of clay stemming is used with slow-burning explosives.

moist place until it undergoes a change in character, is no longer to be considered a permissible explosive. If used in excess of the quantity specified ($1\frac{1}{2}$ pounds), it is not, when so used, a permissible explosive. And when the other conditions have been met, it is not a permissible explosive if fired with a detonator of less efficiency than that prescribed.

Moreover, even when all the prescribed conditions have been met, no permissible explosive should necessarily be considered as *permanently* being a permissible explosive, but any permissible explosive, when used under the prescribed conditions, may properly continue to be considered a permissible explosive until notice of its withdrawal or removal from the list has been officially published, or until its name is omitted from a later list published by the Bureau of Mines.

Furthermore, the manufacturers of a permissible explosive may withdraw it at any time when introducing a new explosive of superior qualities. And after further experiments and conferences the Bureau of Mines may find it advisable to adopt additional and more severe tests to which all permissible explosives may be subjected, in the hope that through the use of such explosives only as may pass the more severe tests, the lives of miners may be better safeguarded.



PUBLICATIONS ON INVESTIGATIONS OF EXPLOSIVES.

A limited supply of the following publications of the Bureau of Mines has been printed and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 15. Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 197 pp., 7 pls., 5 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1914. 50 pp., 3 pls., 7 figs.

BULLETIN 59. Investigations of detonators and electric detonators, by Clarence Hall and S. P. Howell. 1913. 73 pp., 7 pls., 5 figs.

BULLETIN 66. Tests of permissible explosives, by Clarence Hall and S. P. Howell. 1913. 313 pp., 1 pl., 6 figs.

BULLETIN 80. A primer on explosives for metal miners and quarrymen, by C. E. Munroe and Clarence Hall. 1915. 125 pp., 51 pls., 17 figs.

BULLETIN 96. The analysis of permissible explosives, by C. G. Storm. 1916. 88 pp., 3 pls., 7 figs.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 12. The behavior of nitroglycerin when heated, by W. O. Snelling and C. G. Storm. 1912. 14 pp., 1 pl., 2 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 52. Permissible explosives tested prior to March 1, 1913, by Clarence Hall. 1913. 11 pp.

TECHNICAL PAPER 63. Production of explosives in the United States during the calendar year 1912, compiled by A. H. Fay. 1914. 8 pp.

TECHNICAL PAPER 71. Permissible explosives tested prior to January 1, 1914, by Clarence Hall. 1914. 12 pp.

TECHNICAL PAPER 78. Specific-gravity separation applied to the analysis of mining explosives, by C. G. Storm and A. L. Hyde. 1914. 14 pp.

TECHNICAL PAPER 89. Coal-tar products and the possibility of increasing their manufacture in the United States, by H. C. Porter, with a chapter on coal-tar products used in explosives, by C. G. Storm. 1915. 21 pp.

TECHNICAL PAPER 100. Permissible explosives tested prior to March 1, 1915, by S. P. Howell. 1915. 16 pp.

TECHNICAL PAPER 125. The sand test for determining the strength of detonators, by C. G. Storm and W. C. Cope. 1916. 68 pp., 2 pls., 5 figs.

TECHNICAL PAPER 145. Sensitiveness to detonation of trinitrotoluene and tetranitromethylanilin, by G. B. Taylor and W. C. Cope. 1916. 11 pp.

TECHNICAL PAPER 146. The nitration of toluene, by E. J. Hoffman. 1916. 32 pp.

TECHNICAL PAPER 159. Production of explosives in the United States during the calendar year 1915, with notes on coal-mine accidents due to explosives, and a list of permissible explosives, lamps, and motors tested prior to June 1, 1916, compiled by A. H. Fay. 1916. 24 pp.

MINERS' CIRCULAR 7. Use and misuse of explosives in coal mining, by J. J. Rutledge, with a preface by J. A. Holmes. 1913. 52 pp., 8 figs.

MINERS' CIRCULAR 19. The prevention of accidents from explosives in metal mining, by Edwin Higgins. 1914. 16 pp., 11 figs.

PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPER-
INTENDENT OF DOCUMENTS.

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs. 10 cents.

BULLETIN 51. The analysis of black powder and dynamite, by W. O. Snelling and C. G. Storm. 1913. 80 pp., 5 pls., 5 figs. 10 cents.

TECHNICAL PAPER 85. Production of explosives in the United States during the calendar year 1913, compiled by A. H. Fay. 1914. 15 pp. 5 cents.

TECHNICAL PAPER 107. Production of explosives in the United States during the calendar year 1914, with notes on coal-mine accidents due to explosives, compiled by A. H. Fay. 1915. 16 pp. 5 cents.

MINERS' CIRCULAR 6.—Permissible explosives tested prior to January 1, 1912, and precautions to be observed in their use, by Clarence Hall. 1912. 20 pp. 5 cents.

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Bulletin 138

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FRANKLIN K. LANE, SECRETARY

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VAN. H. MANNING, DIRECTOR

COKING OF ILLINOIS COALS

BY

F. K. OVITZ

ILLINOIS COAL-MINING INVESTIGATIONS
COOPERATIVE AGREEMENT

[This report was prepared under a cooperative agreement
with the Illinois State Geological Survey and the department
of mining engineering of the University of Illinois]



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COKING OF ILLINOIS COALS.

By F. K. OVITZ.

INTRODUCTION.

In its endeavor to promote a more efficient use of coal the Bureau of Mines, in cooperation with the Illinois State geological survey and the University of Illinois, has undertaken an investigation of the coking of Illinois coals. The investigation was started by collecting from various sources the data regarding experiments already made by others; these data have been compiled and are presented herein.

The present methods of using coal seem wasteful. When coal is burned in furnaces for the generation of steam, only the heat generated by the fuel is utilized; possible by-products are not only lost but frequently are the source of the smoke nuisance. The volatile matter containing the tar and heavy hydrocarbon gases is difficult to burn completely in many furnaces, and its decomposition produces black smoke. When coal is coked in a by-product oven, the volatile matter is no longer a nuisance but, on the contrary, is the source of many by-products, including benzol, toluol, gas, tar, ammonia, and cyanogen, all of which increase the value that may be obtained from coal.

The coke that remains after the volatile matter has been driven off is a valuable fuel that can be used for almost all purposes for which bituminous coal can be used, and it is necessary for some metallurgical work. It has about the same heating value as anthracite coal. Because of its cleanness and its burning without smoke, it is well suited for domestic heating and the generation of steam in plants where smokelessness is demanded. The use of coke for these purposes would do much to eliminate smoke in cities. The best grades are regarded as essential in many metallurgical processes for reducing ores to their metals.

By-product coking in the United States during the past three years has increased greatly. When the ovens now in course of construction are completed, the total capacity of by-product ovens will be more than double what it was in 1913. This increase has led to a search for coals other than those now known to be available for use in by-product ovens. Tests with coals from the southern part of Illinois

have indicated that coke from Illinois coal unmixed with other coals can be used for fuel purposes and that coke from mixtures of Illinois coal with low-volatile coking coal might be suitable for some metallurgical work. In the tests the yield of ammonia was larger than is obtained from Eastern^a coal.

Evidently the development of methods by which good coke can be made from Illinois coals and the by-products recovered is a matter of high importance. Any investigation that will indicate how, by proper treatment, these coals can be made available to supply the market for coke and gas in the Middle West will increase efficiency in their use, and will aid in conserving the country's supply of high-grade coking coal.

SCOPE OF REPORT.

The first part of this report outlines the present factors in the problem of coking Illinois coals and points out the future prospects. The quality of coke from Illinois coals alone, and from mixtures of Illinois coals with low-volatile coals is described, and the uses for which the coke is suitable are discussed. The gas-making properties and the value of the coals for making by-products are noted. Desirable methods of preparation of the coals, the impurities in them, and the effect of the impurities on the value of the coals for coke and gas making are considered.

The second part of the report deals with the character of Illinois coals, their nature, physical properties, and chemical composition. The nonhomogenous structure of the coal, the variation in composition of different beds, and even of the same bed in different localities are pointed out. Those districts in which the coals contain the smaller amounts of impurities and are more favorable for coking are grouped together.

The last part describes the tests made and gives detailed results. The tests are considered under three classes—those with beehive ovens, those with by-product ovens, and those with gas retorts. The details are as complete as the material available permits. The results of a few tests of the coke for furnace or other use are given.

ACKNOWLEDGMENTS.

Work on this investigation was started by H. C. Porter, formerly chemist of the Bureau of Mines, who made a preliminary survey of the field in order to ascertain if sufficient work had been done to warrant collection of available data.

Especial acknowledgment is given to the companies and individuals who generously furnished valuable data in their possession. Without

^a The term "Eastern coal" as used in this bulletin refers to bituminous coal from the northern part of the Appalachian field.

their cooperation the collection of these results would not have been possible.

Results have also been taken from Government and State publications and from articles in technical journals. For various reasons it has not been considered advisable to refer to every source of information, but reference is made to facts previously published.

Thanks are due to O. P. Hood, chief mechanical engineer of the Bureau of Mines, for his practical advice, and to H. I. Smith, assistant mining engineer of the bureau, who suggested sources of information and otherwise aided in the work. Thanks are also extended to F. W. De Wolf, director of the Illinois Geological Survey, and to S. W. Parr and H. H. Stoek, professors in the University of Illinois, for information on Illinois coals and assistance in obtaining data.

DIVISION OF THE STATE INTO COAL DISTRICTS.

For the purpose of designating and grouping the coals, the State has been divided into eight districts as shown in figure 1. The districting adopted by the Illinois Coal Mining Investigations,^a based on geographical location of the coals and on their physical and geological condition, is used.

The source of the coal is given by county, district, and bed, allowing a general grouping of the results of the tests without giving the names of the mines from which coal was taken. Although this method of designation seems to show that many of the tests cover the same coal, they actually represent different mines working the same bed in different parts of a county or district. In some tests, as those by the United States Geological Survey, the location of the mine is given as it has previously been published. The numbers designating the beds are those used by the Illinois Geological Survey.

The districts shown in figure 1 do not contain quite all the mines worked in the State, but 98.3 per cent of the tonnage is produced in them. In 1912 districts 7, 6, 4, and 5, ranking in the order named, produced 82 per cent of the coal of the State. Districts 2 and 3 each produced less than 1 per cent.

CHARACTER OF COKING PROCESS.

The coking of coal is a destructive distillation either in the absence of air or in the presence of a limited supply. In commercial practice coking is done in beehive and by-product ovens and in gas retorts. The methods of operation differ, as do the products obtained. Beehive and most by-product ovens are for producing metallurgical coke, whereas gas retorts are for obtaining illuminating gas. Coke from gas retorts is not as good as from coking ovens and is usually sold for domestic fuel.

^a Andros, S. O., Coal mining in Illinois: Illinois Coal Mining Investigations Bull. 13, 1915, p. 50.

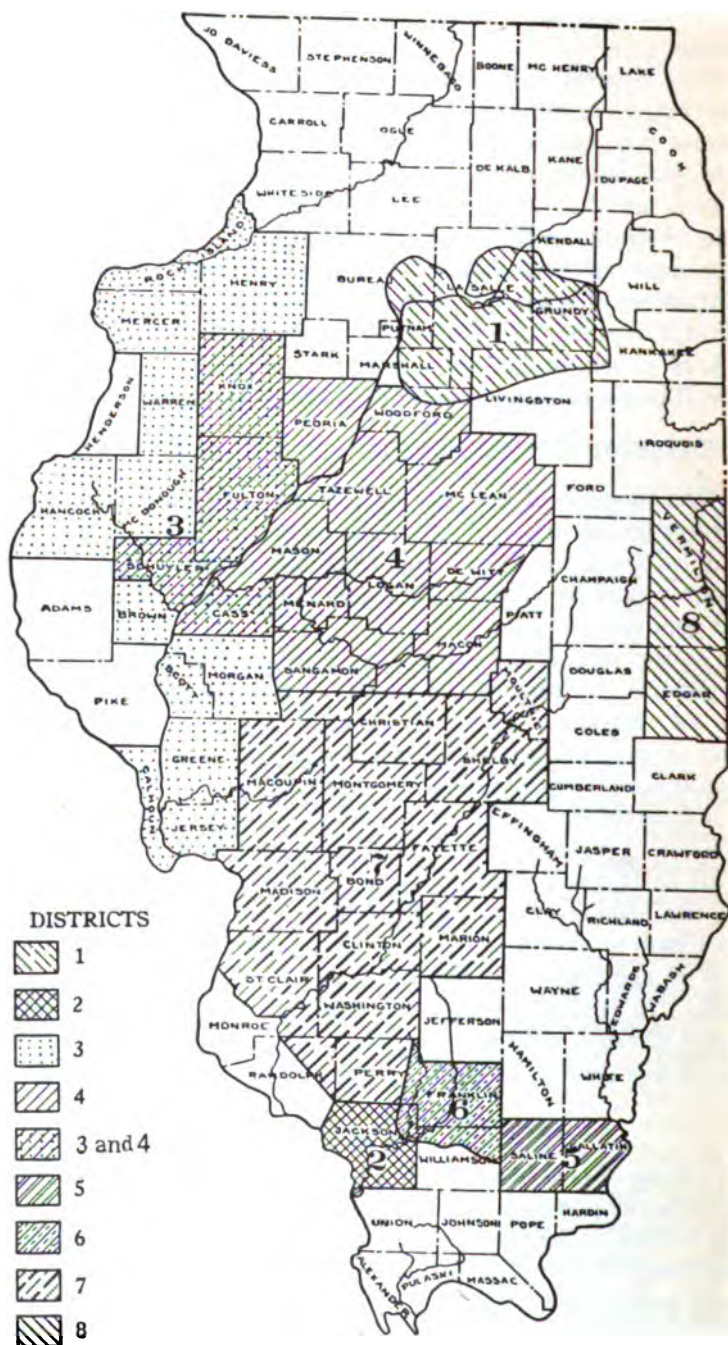


FIGURE 1.—Map of Illinois, showing coal districts adopted under the cooperative agreement.

In beehive ovens coke is ordinarily the only product obtained. The oven is heated from the inside by the combustion of gases driven from the coal and of part of the fixed carbon, a limited supply of air for this purpose being admitted through an opening in the side of the oven. A large share of the volatile matter is burned inside of the oven and the remainder, which passes out through the top, is burned when it comes in contact with air. Thus all the by-products are lost.

In by-product ovens coke is usually the principal product; gas, tar, ammonia, cyanides, benzol, and toluol can be recovered as by-products. The ovens are heated from outside by combustion of part of the gas obtained in coking. No oxygen comes into contact with the volatile matter to burn it and destroy the by-products, and the latter are separated from one another in a suitable recovery system.

In gas retorts illuminating gas is the principal product, coke and ammonia being by-products. The light oil, containing benzol and toluol, is left in the gas to increase its candlepower. The retorts are heated from the outside by combustion of part of the coke produced in the process or of coal. The units are of much smaller size than those used in coking ovens.

PRESENT CONDITIONS REGARDING COKING OF ILLINOIS COALS AND FUTURE PROSPECTS.

EXTENT OF COKE PRODUCTION.

So far as known, commercial coke is not at present being made from Illinois coal in either beehive or by-product ovens, although a small amount is made in gas retorts as a by-product from the manufacture of illuminating gas. Attempts to make commercial coke from Illinois coal in beehive ovens were abandoned, probably because the product could not compete with coke from other coals, and attempts to coke Illinois coal in by-product ovens have been experimental. The results indicate that with by-product ovens coke for fuel can be made from Illinois coal alone and that coke for some kinds of metallurgical work might be made from mixtures of Illinois coal with low-volatile coking coal. At some small gas plants Illinois coal alone is used in retorts for making gas; at other plants one-third Illinois coal and two-thirds Eastern gas coal are used, the gas from the two coals being mixed and the two grades of coke being kept separate.

QUALITY OF COKE.

In general, Illinois coke is light. The cells are not large, but are numerous enough to give a fairly high percentage of porosity. As the cell structure is irregular and the cell walls are thin, the coke is friable. When it is handled Illinois coke tends to shatter and form a

large amount of breeze. Usually the coke comes from the oven in pieces of medium size which have a tendency to break into long fingerlike fragments and to become cross fractured. A characteristic feature is the presence of considerable nonfused material. The percentages of ash and sulphur are usually high. Available tests indicate that for a given purpose more coke from Illinois coal will be required than if coke from Eastern coal is used. The quality of coke made from Illinois coal is improved by mixing low-volatile coking coal with it.

METHOD OF JUDGING QUALITY.

The value of the coke from some of the coals was tested for certain purposes, but the coke from the majority of samples was judged by appearance and physical properties. As the relation between the results of physical tests of coke and its value in practical use is indefinite, different observers may differ widely in their opinions as to the suitability of a given coke for any purpose. Usually the tests made were conducted under operating conditions that gave satisfactory results with other coke, and without regard to any difference in the nature of the Illinois coke. The quantity of coke available was usually too small to permit tests to determine how far changes in practice would effect economy. If through the knowledge gained by experience methods of use could be changed to suit the coke, better results would probably be obtained.

COKE FROM BEEHIVE OVENS.

As is described in detail in a subsequent section, fairly good coke was produced in beehive ovens from some of the coals, whereas the coke from others was poor, and a few samples did not coke at all. The best cokes were light gray and silvery with a good cell structure. The pieces were of fairly large size, but were friable.

A few tests were made to determine the fitness of the coke for metallurgical use. Among these were the cupola tests by the United States Geological Survey ^a which demonstrated that the coke could be used in the foundry, but was not good as coke from Eastern coal. A trial run of about 24 hours was made in a blast furnace at the plant of a large iron and steel company. After four hours the furnace began to slip and became rather cold, and worked in this manner until all of the Illinois coke had been used.

COKE FROM BY-PRODUCT OVENS.

Most of the tests of Illinois coal alone in by-product ovens were made with the coals that had given the most promising results in beehive ovens. Therefore the quality of the coke from different

^a Report of the United States fuel-testing plant at St. Louis, Mo., Jan. 1, 1906, to June 30, 1907: U. S. Geol. Survey Bull. 332, 1908, pp. 94, 113.

by-product oven tests did not vary so much as did the coke from beehive-oven tests. The coke was not as strong as by-product coke from Eastern coals. The structure was fairly uniform, the cells were small and numerous, and the cell walls were thin. The coke had a tendency to break into pieces about 6 inches long and 1 or 2 inches in diameter. Some of it showed numerous cross fractures. When tumbled in a pile it gave a rustling sound and did not have a metallic ring. It contained considerable nonfused material.

Coke made in by-product ovens from mixtures of Illinois coals with low-volatile coking coals was of better quality than that made from Illinois coals alone. The pieces were blocky, the coke was less friable, and the tendency to cross fracturing was somewhat reduced. Mixtures containing 20 per cent of Illinois coal gave better results than mixtures containing larger percentages.

USE OF ILLINOIS COKE IN METALLURGY.

Opinions as to the value of Illinois coke for metallurgical work differ. Most observers agree that it is not suited for use in large blast furnaces, but some think that it can be used in small furnaces, foundries, and smelters. Because of the cell walls being thin, it will not withstand the action of carbon dioxide nor bear the furnace burden as well as coke from Eastern coal. It has been used to some extent during periods when furnaces were slack, but without regard to efficiency, being consumed chiefly to maintain the supply of blast-furnace gas for power. A trial run made in a copper furnace demonstrated that the coke would carry the burden in a furnace of that type.

If absolutely necessary this coke could be used for some metallurgical purposes. It might be used at the present time if it could be sold at a considerably lower price than metallurgical coke from Eastern coal. Just what the difference in price must be can be determined only by a furnace test covering a considerable period of time. However, there are difficulties in the way of any considerable price reduction. The cost of mining coal is about the same as that of mining Eastern coal that makes highly satisfactory coke, but most Illinois coal requires washing, which adds to the cost.

DOMESTIC USE OF ILLINOIS COKE.

The most promising outlet for Illinois coke is as a fuel for domestic and other purposes not requiring coke of as high a grade as is necessary for metallurgical work. Thus, coke that will stand handling without too much loss by breakage can be used for practically all heating and power purposes for which bituminous coal is now used. Furthermore, a coke containing 2 per cent of sulphur will find as ready a sale for such purposes as one containing 1 per cent. The development of this trade to consume the output of several ovens would require

considerable educational work, but once established there would be smaller fluctuations in the demand from year to year than in that for metallurgical coke, the consumption of which is dependent on the iron and steel industry. Producers of coke, by offering it at a reduced price, can induce many consumers of coke for domestic purposes to lay in a supply during the summer months and thus equalize demand. Work has already been started to develop the domestic trade in the Middle West and in the Northwest, with promising results. The high price of the domestic sizes of anthracite and the increasing opposition to the smoke nuisance are factors favoring the use of coke. With the enforcement of smoke ordinances in large cities like Chicago, the railroads will be forced to use coke on locomotives within the city limits or resort to electric power.

The importance of coal gas for lighting has increased recently because of the higher cost of making enriched water gas, caused by a higher price of the oil used in its manufacture. The oil problem is becoming serious with water-gas manufacturers and may cause a return to the use of coal gas. In that event the use of Illinois coals in by-product ovens in certain localities, with sale of the coke to the fuel trade, use of the gas for cooking and lighting, and recovery of by-products, would offer a promising field of development.

USE OF COAL IN GAS RETORTS.

At present a small amount of Illinois coal is used alone for making gas in some of the small plants where the cost of Eastern gas coal is too high. In other plants one-third of the retorts are charged with Illinois coal and the other two-thirds are charged with Eastern gas coal, the coke being kept separate.

Illinois gas coke is of poorer quality than gas coke from Eastern coals; it is light and porous, will not stand much handling, and is easily distinguished from Pittsburgh coke. Lengthening the time of carbonization seems to improve it somewhat. In places where competition is not keen it is sold to domestic trade and where competition is active it is used for heating retorts and for making water gas. In tests of Illinois coke with an 11-foot water-gas machine, owing to the formation of clinkers the capacity was reduced about 20 per cent in comparison with coke from Pittsburgh coal. The consumption of Illinois coke was about 10 per cent larger.

A yield of 4 to 4½ cubic feet of gas per pound of Illinois coal can be obtained, as compared with 5 or 5½ feet from Eastern coal. Gas from Illinois coal has a heating value of about 600 B. t. u. per cubic foot, and a candlepower of about 15. A larger quantity of gas can be obtained, but the excess is mostly hydrogen, which has no illuminating power and a low heating value per cubic foot. As city ordinances or State laws require illuminating gas to have a certain candle-

power or heating value, it is advisable to stop carbonization before all of the lean gas is driven off. The low yield per pound increases the cost of manufacture, as more coal must be handled and a larger generating capacity provided to make the same amount of gas. As coal retorts are usually worked at their full capacity, any increase in output must be obtained from additional retorts, involving a larger investment.

If the quality and appearance of the coke could be improved, the author believes more Illinois coal would be used for making gas. The same coal makes better coke in by-product ovens than it does in gas retorts and at least part of the difference is due to methods of operation. If the coal were finely pulverized before charging, if the size of the charge were increased, and if the method of heating were similar to that used in heating by-product ovens, it seems possible that the quality of the coke could be improved without impairing the yield of gas.

GENERAL RESULTS OF USE OF MIXTURES OF ILLINOIS AND OTHER COALS FOR MAKING COKE.

The mixtures tested in by-product ovens ranged from 20 to 90 per cent Illinois coal with different varieties of low-volatile coking coal. In gas retorts results were obtained with the use of one-third Illinois coals in combination with several Eastern gas coals. The use of 20 to 30 per cent of Illinois coal mixed with a low-volatile coal for making metallurgical coke offers promise of success. Interest in this subject is active at present and the question of whether the use of such coke is profitable may be definitely answered in the near future.

RESULTS IN BY-PRODUCT OVENS.

Mixtures of low-volatile coking coal and Illinois coals make better coke than Illinois coals alone. The effect of the addition of 80 per cent low-volatile coal is well illustrated in Plates I, B, III, B, V, VI, VII, VIII, IX, and X.

The cell walls are stronger and there is less tendency to fingering and cross fracturing than in coke made from Illinois coals only. Compared to coke made from mixtures of Eastern coals it may have an advantage in one respect of Eastern coals—the cell walls being somewhat thinner a relatively larger percentage of surface is exposed to the action of oxygen, so that the coke burns more rapidly, the heat is generated in a small area, and the furnace drives faster. This property is desirable if combined with cell-wall strength sufficient to withstand the burden in the furnace. However, in general the addition of Illinois coal to Eastern coal should be considered an adulteration, and the question is, How much can be added without

reducing the quality of the coke to the point where it can not be used for metallurgical work?

Coke from mixtures of 20 per cent Illinois coal and 80 per cent low-volatile coal has been used in blast furnaces for several consecutive weeks. No great trouble was experienced although there were some soft spots in the coke. As far as furnace operation is concerned, coke from such a mixture can be used. Possibly the proportion of Illinois coals can be increased to 30 per cent or more, but the use of even 20 per cent in all the by-product ovens near the Illinois coal field would mean a large annual consumption. There are more than 1,500 by-product ovens, with a capacity of about 9,000,000 tons of coal annually, within a short distance of this field.

Naturally the principal incentive to use such a mixture would be a reduction in the cost of fuel for metallurgical work. Illinois coal is cheaper in the Middle West than the Eastern high-volatile coals it would replace, and, therefore, other factors being the same, coke could be made at a lower cost per ton. If the same amount of this coke as of coke from Eastern coals were required per ton of iron, the saving from the use of Illinois coal could be calculated. However, both known and unknown factors enter into the problem and as yet sufficient information has not been obtained to show just how much saving would result. The yield of coke is lower, and more coke is required per ton of iron produced, but to offset these disadvantages a larger quantity of by-products and more blast-furnace gas would be made. If it is shown that the use of coke made from mixtures containing 20 per cent of Illinois coal means a substantial saving, a considerable amount of such coke undoubtedly will be used.

By adding a low-sulphur Eastern coal to Illinois coal, coke of satisfactory sulphur content can be obtained. If 80 per cent of Eastern coal containing 0.7 per cent of sulphur is used, the 20 per cent of Illinois coal can contain about 2 per cent of sulphur without the sulphur content of the mixture running over 1 per cent. A large amount of Illinois coal contains less than 2 per cent of sulphur and hence would be available for use in such mixtures.

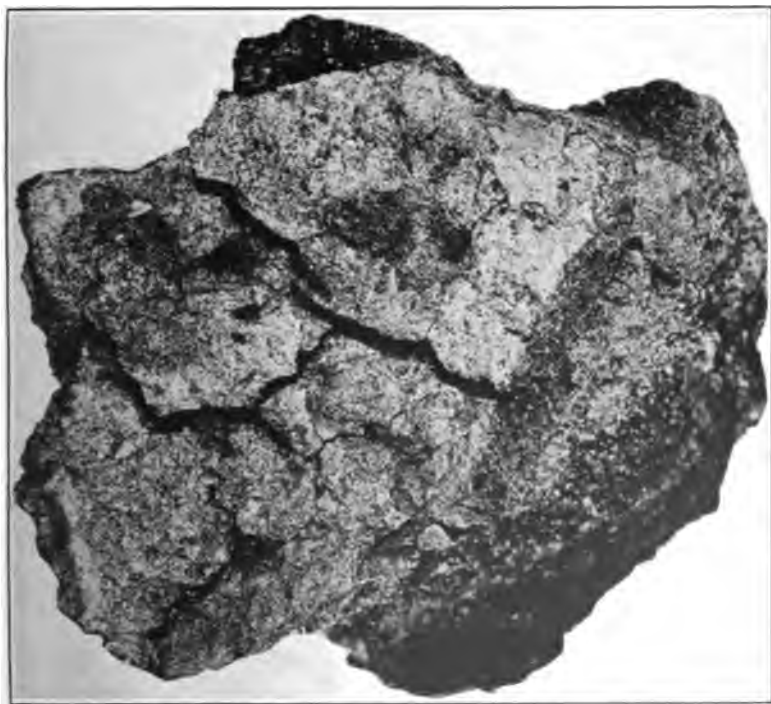
RESULTS IN GAS RETORTS.

The results obtained in gas retorts from the use of Illinois and Eastern coals in separate retorts have caused several plants to adopt such procedure. The plants have been able to show a reduction in cost of gas of about 2 cents per 1,000 cubic feet when one-third Illinois and two-thirds Eastern gas coal were used.

The gas and the by-products from the two sets of retorts go into the same recovery system, but the coke is kept separate. The quality of the gas is about the same as from Eastern coal alone and the yield is only slightly less. There is not much difference in the



A. LUMP OF SOUTHERN ILLINOIS COAL, FROM FRANKLIN COUNTY. SHOWS ALTERNATING LAYERS OF DULL AND SHINY APPEARANCE.



B. COKE FROM 20 PER CENT ILLINOIS (DISTRICT 6, BED 6) COAL AND 80 PER CENT LOW-VOLATILE COAL, TEST 10. BLOCKY PIECES, GOOD COLOR.

yields of tar and the ammonia, the former being slightly less and the latter slightly more than from Eastern coal alone. The coke from Illinois coal is used for heating retorts and making water gas, and that from the Eastern coal is sold for domestic use. If the coals were pulverized and thoroughly mixed instead of being charged into separate retorts, it is believed that all of the coke could be made suitable for the domestic trade. The cost of such preparation might be met by increasing the proportion of the cheaper Illinois coal.

EFFECTS OF VARIOUS FACTORS ON COKING OF ILLINOIS COALS.

USE OF HIGH TEMPERATURES.

There is a rather general opinion that high temperatures give better results in making coke from Illinois coal. Some investigators believe that high temperature combined with fine grinding is all that is necessary to produce from Illinois coal coke suitable for any metallurgical use. This possibility remains to be demonstrated, for the exact improvement by high temperature can be determined only by thorough investigation. The results of tests herein reported seem to indicate that high temperature does improve the quality, but the results are not complete enough for drawing a definite conclusion.

ADDITION OF PITCH.

Although no results of tests of making coke from Illinois coals with the addition of pitch are reported, a few such tests have been made. Adding about 5 per cent improves the coke. The cell structure is more open, and the tendency to cross-fracturing is reduced.

The addition of pitch to low-grade coking coals may furnish a profitable use for any excess supply of pitch resulting from the increased number of by-product ovens and may develop a market for a large quantity. The field invites investigation.

YIELD OF BY-PRODUCTS.

The yield of gas from by-product oven tests ranged from 11,000 to 13,000 cubic feet per ton (2,000 pounds) of coal, and in by-product laboratory tests averaged about 10,500 cubic feet per ton. Thirteen thousand feet is a high yield and probably could not be obtained in regular practice. The yield of gas from Illinois coals is slightly smaller than from high-volatile Eastern coals, but is larger than from low-volatile coals. When 20 or 30 per cent of Illinois coals are mixed with low-volatile coals the yield is about the same as from similar mixtures of Eastern coals.

More ammonia is obtained from Illinois coals than from Eastern coals. The maximum yield in by-product ovens was 34 pounds of

ammonium sulphate per ton of coal, and the average was 26.6 pounds, equivalent to 6.8 pounds of ammonia (NH_3). The maximum yield by laboratory test was 41 pounds of ammonium sulphate per ton, and the average was 34.9 pounds. In gas retorts the high yield was 24.9 pounds of ammonium sulphate or 6.42 pounds of ammonia (NH_3). It is safe to say that a yield of 6 pounds of ammonia, or more than 23 pounds of ammonium sulphate, could be obtained in the regular operation of gas retorts. In some of the gas tests the quantity of ammonia was determined at the outlet of the exhauster and therefore is low, as considerable ammonia is absorbed before the gas reaches the exhauster.

The average yield of tar from by-product oven tests was 11.9 gallons per ton of coal, and by laboratory test was 7 gallons. Although few data were obtained for the tar from gas retorts, the yield is probably 10 to 12 gallons per ton, compared with about 15 gallons from Pittsburgh coal. The quality depends upon temperature and the consequent amount of cracking; in general, the tar from the Illinois coal compares well with other tars.

The laboratory tests gave an average yield of 18.9 pounds, or slightly less than 3 gallons, of light oil per ton of coal. Only laboratory determinations were made, and no results of oven tests are available for comparison. Consequently, it is difficult to predict how well the laboratory results will be confirmed in practice. At low temperatures the aromatic hydrocarbons are not formed to any great degree, and at high temperatures they decompose rapidly. Therefore, the laboratory test must reproduce closely the amount of cracking in the ovens in order to be of practical value.

EFFECT OF IMPURITIES IN COAL.

High sulphur and ash contents limit the use of Illinois coals for coking, as both coke and gas manufacturers desire coals as low as possible in sulphur. The customary specification for metallurgical coke is not more than 1 per cent of sulphur, which means that the coal can not contain much more than 1 per cent, as about half of the sulphur remains in the coke and the yield of coke, by weight, is 60 to 70 per cent. Sulphur must be removed from illuminating gas, and the more that must be removed the higher is the cost of purification. For this reason high-sulphur coal will not be used when low-sulphur coal can be obtained.

All of the ash in a coal remains in the coke. If a coal contains 10 per cent of ash and yields 60 per cent of coke, the coke will contain 16.6 per cent of ash. Ash does not fuse at coking temperatures; it increases the amount of nonfusible material that must be cemented by the fusible part of the coal and makes the fusion less uniform.

NEED OF WASHING ILLINOIS COALS.^a

Most Illinois coals should be washed before coking in order to reduce the sulphur and the ash contents. A limited amount of Illinois coal has a sulphur content of about 1 per cent, but the average is higher. The coal from district 2, which contains less sulphur and ash than that from any other district, averages 1.29 per cent sulphur and 5.72 per cent ash. Ash contents of the coals from all the districts are given in Table 1 (p. 23). About 25 per cent of the sulphur and 20 to 30 per cent of the ash can be removed by washing or dry-cleaning. Part of the sulphur combined as iron pyrites can be removed, the proportion depending on whether the pyrite is in lumps or is disseminated in fine particles. In the latter form it can not be removed by washing, nor can organic sulphur be removed in that way.

Proper sizing of the coal is important. The impurities in the very fine coal can not be reduced much and for this reason the fines should be removed before washing. In selected coals the sulphur can be reduced to less than 1.5 per cent, and when these are used in mixture with low-sulphur Eastern coals, coke can be produced low enough in sulphur to meet metallurgical requirements.

Part of the ash is intimately mixed with the coal substance, is an inherent part of the coal, and can not be removed by cleaning. The remainder is from layers or seams of pyrites, shale, or other materials in the coal bed, which were mixed with the coal during mining. This part can be reduced by care in mining and by subsequent cleaning.

The cost of washing is 5 to 10 cents per ton, and 10 to 25 per cent of the coal is lost during the operation. The increase in the price of coal due to washing is shown by the following example. If the cost of washing is 5 cents per ton, and 15 per cent of the coal is lost, then with raw coal at \$1 per ton the price of the washed coal would be as follows:

$$\frac{1.00+0.05}{0.85}=\$1.23 \text{ per ton.}$$

EFFECT OF CRUSHING COAL.

Tests in beehive ovens were made with both coarse and fine coal, and invariably the fine coal produced the better quality of coke. The reason is probably found in the nonhomogeneous character of Illinois coal. It is made up of layers, some of which fuse, whereas others do not, the nonfusible layers causing the weak spots in the coke. Fine crushing breaks up the nonfusible seams and pieces of slate so that they can be evenly mixed through the mass of coal. Thus, the small particles of nonfusible material are brought into close contact with fusible particles, become cemented during coking,

^a For results of experiments in washing Illinois coal, see Lincoln, F. C., Coal washing in Illinois: Univ. of Illinois Eng. Exp. Sta. Bull. 69, 1913, 106 pp.

and the coke is stronger and more uniform. Large pieces of unfused material give more chance for the coke breaking into small pieces. If there is too much nonfusible material, the particles will not be cemented and the coke will be granular. In order to get the best results in coking Illinois coals they should be crushed very fine.

If more than one variety of coal is used, proper crushing is necessary, so that the different coals can be thoroughly mixed; otherwise the coke will not be uniform.

COKING PROPERTY OF ILLINOIS COALS.

Practically all Illinois coals will fuse to some degree. Just what material imparts the coking property is not known, and many explanations have been advanced. These have been based on different considerations—the original material from which the coal was formed, the process of coal formation, and the chemical composition and the physical character of the coal. Because of the extremely complex nature of coal and the difficulty of determining and identifying constituents, the real cause remains unknown and the explanations advanced are largely hypothetical.

Chemical analysis does not show whether a coal will make good coke. Noncoking and coking coals may have practically the same proximate and ultimate composition by analysis and yet the relative amounts of resinous, humic, and cellulosic materials may differ decidedly. Coals that coke contain a considerable amount of resinous material and have a relatively high percentage of available hydrogen and a low oxygen content. However, these characteristics indicate only that a coal may coke, and can not be relied on as a measure of coking quality. In the present state of our knowledge of coking the only safe method of determining what a given coal will do is by actual test in an oven.

ILLINOIS COALS AVAILABLE FOR COKE MAKING.

Most Illinois coals will make coke of some sort. In general, coals from districts 2, 5, and 6 (fig. 1), in the southern part of the State, make better coke than those from other districts. However, not all the coals from the districts mentioned will coke, and the only way of determining the coking property of coal from any mine is by proper test.

District 6 seems to be the most available and promising field in the State. It contains a rather large area of coal and produces about one-fifth of the total output of Illinois. The impurities in the coal from that district are lower than in the coal from the other districts, except district 2; the sulphur averages about 1.5 per cent and the ash considerably less than 10 per cent. Much of the coal is

low enough in sulphur to produce coke that will meet metallurgical requirements for sulphur content, if the coal is mixed with low-sulphur coking coal. The coal is also used for the manufacture of illuminating gas, being used alone and with Eastern gas coals.

The coals from district 5 in composition and coking property are similar to the coals from district 6, but are higher in sulphur, and therefore not as well suited for coking and gas making. District 5 produces about 7 per cent of the coal output of the State.

By analysis the coal from district 2 is of better quality than any other coal in the State, being higher in fixed carbon and lower in ash and sulphur. It is used successfully for gas making in some small plants in Illinois and Missouri, and formerly was coked in beehive ovens in the vicinity of St. Louis. The district is small, produces only about 1 per cent of the coal mined in the State, and hence is not of great importance as a possible source of coking coal.

Tests of coals from other districts of the State show that although some of the coals make fairly good coke, most of them are so high in sulphur that the coke could only be used for fuel or domestic purposes. Some of these coals are equal in fusing property to those of district 6.

The work that has been done on coking Illinois coals should be regarded as preliminary. Knowledge of the mechanics of the coking process is still limited, and it is entirely possible that some comparatively small improvement in the process may give the proper combination to produce good coke from Illinois coals.

CHARACTER OF ILLINOIS COALS.

GEOLOGIC FEATURES.

Illinois coals are classed as bituminous and in degree of maturity are regarded as intermediate between the Eastern bituminous and the Western subbituminous coals. The productive beds are found in the Pennsylvanian series of the Carboniferous system. Beds 1, 2, 5, 6, and 7 are mined commercially, and are described in publications of the Illinois Geological Survey.^a

PHYSICAL CHARACTERISTICS.

Illinois coals look like other bituminous coals, being composed of alternating shiny and dull layers. They are less friable than Pocahontas or New River coals and stand shipping fairly well. The fractures, although sometimes conchoidal, are usually along horizontal and vertical planes more or less at right angles to each other, so that the coals have a tendency to break into cubical pieces. On an average about 15 per cent of the total output is in lumps that pass over a 6-inch screen, and about 20 per cent will pass through a $\frac{1}{4}$ -inch screen.

^aSee Andros, S. O., Coal mining in Illinois: Illinois Coal Mining Investigations Bull. 13, 1915, 250 pp.

The Illinois coals are usually classed as free burning and noncoking, but when subjected to destructive distillation fuse into more or less coherent masses. Practically all of the coals fuse to some degree. The coke from different coals varies somewhat, but, as stated, is generally light and friable.

The structure is shown in Plates I, A, and II. Plate I, A, showing a lump of Franklin County coal, indicates clearly its nonhomogeneous nature. Plate II shows a magnified section taken from near the center of the lump shown in Plate I, A. Here the black shiny layers are nearly uniform, but the dull layers are divided into very thin laminæ. This structure is fairly typical of southern Illinois coals.

Reinhardt Thiessen,^a assistant chemist of the Bureau of Mines has examined Illinois coals under the microscope and found that the shiny layers consist of fragments of "wood" and range from a few microns to several millimeters in thickness. The "woody" nature is readily discerned by examining sections across the grain. Most of the "woody" layers are highly resinous, and in some samples almost the entire "woody" component has disappeared, and little is left except the resinous material. The dull layers are generally thicker than the shiny ones and are composed largely of finely disintegrated material, or "débris." On closer examination they are found to be further divided into dull and shiny laminæ a fraction of a millimeter thick. As in the thicker layers, the shiny parts are of a "woody" nature and the dull parts are composed of disintegrated material.

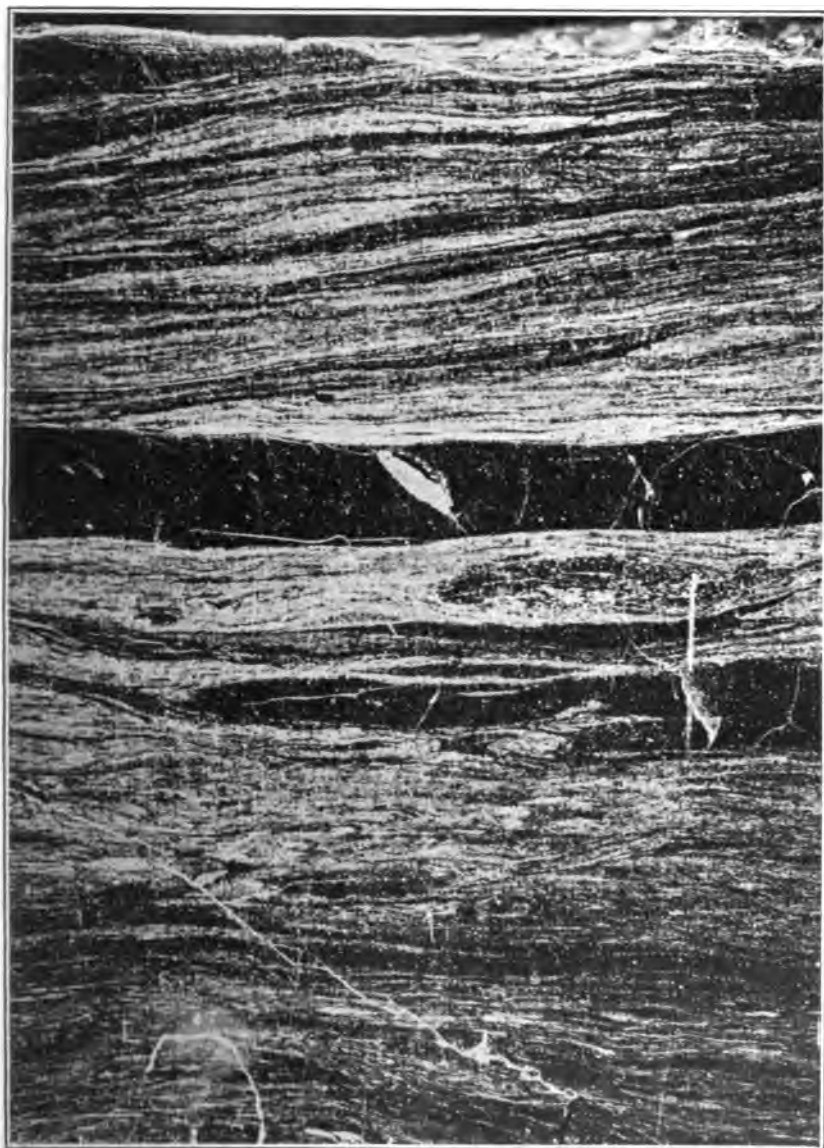
Examination of a partly coked lump shows that the shiny layers possess the coking property and that the dull layers do not. Under the influence of heat the shiny parts fuse and swell, whereas the dull parts simply char and crack. Consequently fine crushing and thorough mixing of the parts that fuse and those that do not seems to be necessary in order to obtain the best results in coking. Further microscopic study of coal may lead to a better understanding of the causes of coking.

CHEMICAL CHARACTERISTICS.

Although Illinois coals are of the same general type, the composition of samples from different parts of the State varies. The proximate analyses of averages are given, by districts, in Table 1.^b In interpreting them it must be remembered that ash and sulphur are incidental impurities in coal and vary locally. In any one mine the proportions of these elements may be quite different from the average for the district. For instance, some mines in district 6 yield coal that contains considerably less than the average of 1.53 per cent of sulphur.

^a See White, David, and Thiessen, Reinhardt, The origin of coal: Bull. 38, Bureau of Mines, 1913, p. 25⁷

^b Andros, S. O., Coal mining in Illinois: Illinois Coal-Mining Investigations Bull. 13, 1911, p. 57.



SECTION FROM NEAR CENTER OF LUMP OF COAL SHOWN IN PLATE I, A, MAGNIFIED 10 TIMES. SHOWS SUBDIVISIONS OF THE DULL LAYERS.

TABLE 1.—Average analyses of Illinois coals arranged by districts.^a

District No.	Seam No.	Number of samples.	Proximate analysis of coal. ^b				Sulphur.	Heating value.	Heating value of unit coal. ^c
			Moisture.	Volatile matter.	Fixed carbon.	Ash.			
			<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>P. ct.</i>	<i>B. t. u.</i>	<i>B. t. u.</i>
1.....	2	33	16.18	38.83	37.89	7.08	2.90	10,981	
			46.38	45.21	8.45	3.45	13,101	14,528	
2.....	2	15	9.28	33.98	51.02	5.72	1.20	12,488	
			37.46	56.24	6.30	1.42	13,765	14,818	
3.....	1	11	15.58	30.17	35.80	9.45	4.00	10,673	
			46.40	42.41	11.19	5.55	12,643	14,546	
	2	3	17.40	33.30	41.48	7.82	2.03	10,811	
			40.32	50.23	9.48	2.45	13,001	14,663	
4.....	5	54	15.10	36.79	37.59	10.53	3.52	10,514	
			43.33	44.28	12.40	4.15	12,384	14,447	
5.....	5	27	6.75	35.49	48.72	9.04	2.92	12,276	
			38.06	52.25	9.69	3.13	13,165	14,812	
6.....	6	58	9.21	34.00	48.08	8.71	1.53	11,825	
			37.45	52.96	9.59	1.68	13,025	14,585	
7.....	6	76	12.56	38.06	39.06	10.33	4.01	9,848	
			43.52	44.67	11.81	4.50	12,406	14,377	
8.....	6	31	14.45	35.88	40.33	9.34	2.55	10,919	
			41.94	47.14	10.92	2.98	12,764	14,567	
	7	18	12.99	38.29	38.75	9.98	2.93	11,143	
			44.01	44.53	11.47	3.37	12,807	14,740	

^a Analyses made by J. M. Lindgren under direction of Prof. S. W. Parr.

^b Upper horizontal line of figures opposite each district number represents coal as received; lower line of figures represents moisture-free (or "dry") coal.

^c "Unit coal" refers to the combustible organic material. The heating value on the unit-coal basis is obtained from the following formula: Unit B. t. u. = $\frac{\text{Dry B. t. u.} - 5,000 S}{1.00 - (1.08A + .55S)}$, in which A is weight of ash per gram, and S is weight of sulphur per gram.

The fixed carbon varies from about 42 to 57 per cent on the dry-coal basis. The proportion of fixed carbon indicates in a general way the yield of coke that may be expected from a coal; that is, the higher the fixed carbon the greater the probable yield of coke. The fusibility and swelling when heated can be judged to some extent by examining the residue from a fixed-carbon determination, but this does not afford a criterion of the result of heating under other conditions.

The volatile matter ranges from about 37.5 to 46.5 per cent on a dry-coal basis. It is the source of gas, tar, ammonia, and other by-products. Parr^a has shown that a large part of the volatile matter in Illinois coal is inert. Forty per cent of the volatile matter, or 14 per cent of the entire coal, is inert, as contrasted with 22 and 4.2 per cent in the case of Pocahontas coal and 47 and 21.63 per cent in the case of North Dakota lignite, and this fact should be given consideration in estimating the value of coal for gas making. When coal is carbonized the inert volatile matter is driven off largely as water of decomposition, carbon dioxide, and nitrogen. These constituents are not combustible and reduce proportionately the quantity of useful gas that can be obtained.

The average sulphur content, by districts, varies from 1.42 to 5.55 per cent on the dry-coal basis. Some mines produce coal containing less than 1 per cent, but they are exceptions. The sulphur is combined less

^a Parr, S. W., Composition and character of Illinois coals: Illinois State Geol. Survey Bull. 3, 1906, 86 pp.

chiefly as iron pyrites (FeS_2); but part of it is combined with the coal substance in organic form. Also a small part is frequently combined as calcium sulphate, present in thin white flakes along the cleavage planes of the coal. When coal is coked approximately one-half of the sulphur is driven off in the volatile matter and one-half remains in the residue. All of that combined as sulphate remains in the coke.

Ash in most Illinois coals is high, averaging about 10 per cent (dry-coal basis). It is largely an incidental impurity and varies in coals from different parts of a district and even in coal from different parts of the same mine. In the coking process all of the ash remains in the coke; therefore coke shows a higher percentage of ash than the coal from which it is made. High ash content is objectionable because the ash replaces an equal amount of combustible in the coke and correspondingly decreases the heating value. However, such increase of ash is more than offset by the removal of inert material. As pointed out on page 23, the inert volatile matter may amount to 14 per cent of the weight of Illinois coal, which, added to the moisture, may increase the noncombustible material, aside from the ash, to 25 per cent of the weight of the coal. Thus the coke is a more concentrated fuel than the raw coal, even though the coke has a higher ash content.

Phosphorus in Illinois coals is usually low. All of the phosphorus in coal is retained in the coke. As far as using the coke for metallurgical purposes is concerned, the proportion of phosphorus in Southern Illinois coal is not objectionable.

GROUPING OF COALS ACCORDING TO COMPOSITION.

In Table 2 following, the coals are arranged according to their fixed carbon content on the moisture-free and ash-free basis, the values for fixed carbon and volatile matter being obtained by calculation from the figures given in Table 1. The values for oxygen and available hydrogen are averages taken from ultimate analyses given in Bulletin 16 of the Illinois State Geological Survey.^a No ultimate analyses could be obtained for samples from districts 2 and 3.

TABLE 2.—Average analyses of Illinois coals arranged according to fixed carbon content on the moisture-free and ash-free basis.

Fixed carbon.	Volatile matter.	Oxygen.	Available hydrogen.	Seam No.	District No.
<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>	<i>Per cent.</i>		
60.0	40.0			2	2
58.5	41.5	10.12	3.98	6	6
57.8	42.2	9.75	4.08	5	5
55.5	44.5			2	3
52.9	47.1	11.41	3.91	6	8
50.7	49.3	11.64	3.76	6	7
50.6	49.4	11.61	4.20	2	1
50.5	49.5	11.42	4.09	5	4
50.3	49.7	11.32	3.98	7	8
47.7	52.3			1	3

^a Parr, S. W., The chemical composition of Illinois coal: State Geol. Survey Bull. 16, 1910, pp. 203-243.

The table shows that fixed carbon and volatile matter vary considerably in samples from different parts of the same bed. For example, the samples from seam 2 in district 2 contained an average of 60 per cent of fixed carbon, whereas samples from the same seam in district 1 contained an average of 50.6 per cent fixed carbon. Hence the seam number is not a reliable guide to the composition and properties of any sample of coal from it. If coal from a certain bed in one locality produces coke, coal from the same bed in another locality will not necessarily produce similar coke.

Coal from districts 2, 5, and 6 contains more fixed carbon and less volatile matter than that from the other districts. There is an average decrease of more than 7 per cent in fixed carbon and a corresponding increase in volatile matter in the coal from these districts as compared with that from districts 1, 4, 7, and 8. Coal from districts 2, 5, and 6 is also lower in oxygen than that from districts 1, 4, 7, and 8. Although no ultimate analyses of coal from district 2 are available, it is quite likely that this coal is also low in oxygen, as the fixed carbon is high.

Districts 2, 5, and 6 comprise an area extending across the southern part of the State, including Jackson, Franklin, Saline, and Gallatin Counties and that part of Perry County east of the Duquoin anticline. The coals are somewhat similar in composition and differ considerably from the other coals of the State, being higher in fixed carbon and lower in oxygen, sulphur, and ash, and hence better suited for coking. These coals are more mature than the other coals of the State; that is, they retain less of their volatile constituents, as shown by the higher percentage of fixed carbon and the lower percentage of oxygen. In these respects they approach nearer to the good coking coals of the East, which are characterized by a relatively high fixed carbon and a low oxygen content. Just what relation exists between oxygen and the coking property is not understood, but seemingly high oxygen content in a coal tends to destroy the coking property. It has been shown on a laboratory scale that coal that has been heated to 120° C. for several hours in an atmosphere containing oxygen tends to become noncoking. It is generally believed that weathered coal produces poorer coke than freshly mined coal, and this belief may be justified by the fact that weathering is largely a slow oxidation of the coal substance.

As regards composition alone, the coals from the southern part of the State—districts 2, 5, and 6—are better suited for coking than the other coals. This statement is based upon the average composition by districts; certain coals in any one district may differ from the average, and therefore would not be included.

HISTORICAL REVIEW OF COKING OF ILLINOIS COALS.

Fairly satisfactory results were obtained in some of the early attempts to coke Illinois coals, and these excited strong hopes that the problem of local coke production had been solved.

TESTS REPORTED BY LUEBBERS.

In 1872 Luebbers^a described a process of making coke from Illinois washed slack on a large scale, stating that although certain experiments under way were in their infancy the results were promising.

The ovens were built in the form of an arched culvert, 1½ to 6 feet wide and closed at either end by cast-iron doors, and the coke was pushed with a ram. The length of the ovens was not stated. The ovens were built in rows of 20 to 30. The coking time was 24 to 36 hours. The coke, which was somewhat porous, but sufficiently dense for use in blast furnaces, weighed about 38 pounds per bushel. The yield was about 60 per cent of the weight of the washed coal. No mention is made of the location of the ovens nor of the source of the coal used.

USE OF BEEHIVE OVENS AT MURPHYSBORO, ILL.

In about 1886 some beehive ovens were operated at Mount Carbon, near Murphysboro, Ill. They produced considerable coke, probably three or four hundred thousand tons. Some of it was used in near-by blast furnaces and some of it for reducing zinc ores in the Missouri district. The ovens were abandoned years ago and are at present in ruins. The coal used was from bed 2 in district 2.

EXPERIMENTS IN SANGAMON COUNTY.

Experiments by an iron company, a coal company, and a gas company are described in an article by Ridgely.^b

The coal company was throwing away more than 500 tons of fine coal daily, and it proposed to coke this coal, from Sangamon County, to recover the tar and ammonia.

An experimental oven was tried, but the coke was not satisfactory. The company then undertook to use the coal in a by-product gas producer and recover ammonia and tar. All of the ammonia could not be recovered. Although as high as 45 pounds of ammonium sulphate per ton was obtained at times, this yield could not be maintained regularly. Altogether, four or five hundred tons of ammo-

^a Luebbers, H. L., The manufacture of coke from Illinois coal: Trans. Am. Soc. Civil Eng., vol. 2, 1874, p. 163.

^b Ridgely, Charles, An experiment with the by-products of coal: Illinois Min. Inst. Jour., vol. 3, 1894, p. 39.

nium sulphate and about 5,000 barrels of tar were obtained. As the production was not economical the work was stopped and has never been resumed.

OVENS AT EQUALITY, ILL.

About 1890, 24 ovens were constructed at Equality, Ill. The ovens were about 20 feet long, 5 feet high, and 20 inches wide. They were built in a row with a flue between each two ovens. Coke was made from coal from district 5, bed 5, for a number of years, but since about 1910 the ovens have been idle. Coke of good quality is said to have been produced, but a high sulphur content restricted its usefulness.

OVENS NEAR SPARTA, ILL.

Some years ago 102 ovens of similar type to those at Equality were erected near Sparta, Ill. It is understood they were built to make coke from slack coal from district 7, bed 6. The ovens were not used much, some of them never being fired. They are still standing but have been idle for years.

LEITER EXPERIMENTS IN SOUTH CHICAGO.

About 1899, Levi Z. and Joseph Leiter supplied the funds to enable experiments to be made by Joseph Heminway in a modified beehive oven invented by him. Several coals were tested in a single oven, with gratifying results, and later 24 ovens were built at Thirty-fourth and Iron Streets, South Chicago, where tests were made on a larger scale.

The ovens differed from the ordinary beehive type in being fitted with a fan for supplying hot or cold air over the surface of the coal. By this arrangement it was possible to obtain a higher temperature and better control in heating the ovens. Claims were made that coke could be produced from coals previously considered noncoking, that a considerable part of the sulphur could be eliminated, and that in 12 to 15 hours coke could be made equal to the usual 48 or 72 hour coke. The ovens could be equipped for saving by-products.

The process was owned by the Universal Fuel Co., which intended to issue licenses to operating companies, but no other ovens were ever built, so far as known. Coals from Ohio, Indiana, Illinois, Iowa, and Kansas were tested, and it was reported the coke made from them was suitable for metallurgical use. After about two years, operation of the ovens at South Chicago was discontinued. A description of the ovens and some of the results obtained have been given by Moss.^a

^a Moss, R. S., Improved Heminway process: *Mines and Minerals*, vol. 21, 1901, p. 412.

OVENS IN ST. CLAIR COUNTY.

About 1902 five or six beehive ovens were built to make coke from St. Clair County coal. It was stated that the ovens were built because a suitable contract price could not be obtained for Eastern coke. After the ovens had been completed a satisfactory price was obtained and they were little used. The coke was used for burning lime and was said to be satisfactory for that purpose.

EXPERIMENTS NEAR HARRISBURG, ILL.

An experimental oven has been erected near Harrisburg, Ill., and various Illinois coals have been tried in it. Although the results have been sufficiently promising to warrant the expenditure of considerable money, and although it is claimed that coke can be made from Illinois coals, at present the process is still in the development stage.

Summers,^a the inventor, believes that good coke can be made in this oven from coal regarded as noncoking. The oven consists of a retort through which the coal is caused to move continuously and a furnace for heating the retort. Coal is charged through a hopper at one end of the retort, and coke is discharged at the other end. The temperature increases from the charging end to the exit end and volatile matter that is driven off at low temperature is forced to pass through hot coal or coke, thereby being decomposed and depositing carbon, which cements the particles.

DETAILED RESULTS OF COKING TESTS OF ILLINOIS COALS.**SCOPE OF TESTS REVIEWED.**

The detailed results of coking tests of Illinois coals presented in the following pages include results of tests at high temperatures in beehive and by-product ovens, in gas retorts, and in the laboratory. The results given include those obtained with Illinois coals alone, with mixtures of 20 to 90 per cent of Illinois coal with low-volatile coals, and with one-third Illinois coals used in combination with two-thirds Eastern gas coal. Some of the records of tests are complete enough to include figures for the yield of by-products, whereas others are incomplete and contain information on a particular part of the process only. In some tests the coke was used in a blast furnace, a cupola, or a water-gas generator, or as fuel for heating retorts and generating steam, but in most cases the quality was judged from physical properties and appearance. It is believed that the results given are sufficient to show, in a general way, the character of the results that have been obtained up to the present time.

The results of the coking of Illinois coals at low temperatures are not included. Low-temperature coking has been investigated by

^a Summers, L. L., U. S. patents 943809 and 943610, Dec. 14, 1909; U. S. patent 951786, Mar. 8, 1910.

Parr and others at the University of Illinois.^a They obtained a solid residue, containing 18 to 22 per cent of volatile matter, which makes a smokeless fuel suitable for domestic purposes. In a suction gas producer it proved fully equal if not superior to anthracite. The tar contains 28 to 30 per cent of tar acids and gives promise of being suitable for wood preservation. Plans are under way for extending the experiments, to allow study of the process on a commercial basis.

DISTRIBUTION OF COALS TESTED.

Coals from districts 1, 4, 5, 6, 7, and 8 (fig. 1) are represented in the results of tests presented. Table 3 gives the number of coals tested in each kind of oven and the district and seam from which each coal was obtained. Most of the tests were made with southern Illinois coals, especially those from district 6. No results of tests of coals from districts 2 and 3 are given. Coal from district 2 is being used successfully for manufacturing illuminating gas and would undoubtedly give as good results as any Illinois coals in by-product ovens. However, the possible output is small and therefore it does not offer much of a field for development.

TABLE 3.—*Distribution of coals tested.*

District No.	Seam No.	Number of coals tested in—		
		By-product ovens.	Beehive ovens.	Gas retorts.
1.....	2	2		
4.....	7	1		
5.....	5	2	1	
6.....	5	5	1	1
6.....	6	32	9	5
7.....	6	1	12	
8.....	6	1		

AVAILABLE DETAILS OF TESTS.

The results of tests have been collected from various sources. The conditions under which they were run were different and it was impossible to obtain details of some of the tests. Methods of sampling and analyzing the products and preparing the coal differed, as did the temperatures of carbonization, and in many instances complete information on these points could not be obtained. The results are not all stated on the same basis. For example, in some of the tests the analysis of coal and coke and the yield of coke are given on the dry basis, and in others they represent coal "as received;" or the figures

^a See Parr, S. W., and Olin, H. L., The coking of coal at low temperatures: Univ. of Illinois Bull. 60, 1912, 46 pp.; Parr, S. W., and Olin, H. L., The coking of coal at low temperatures, with special references to the properties and composition of the products: Univ. of Illinois Bull. 79, 1915, pp. 29-30; Parr, S. W., and Francis, C. K., The modification of Illinois coal by low-temperature distillation: Univ. of Illinois, Bull. 24, 1908, 48 pp.

for gas may not have been reduced to the same conditions of temperature and pressure in all cases. For these reasons care should be used in comparing the figures and too much reliance should not be placed upon the results of a single test.

Observations on the tests are not uniform and information as to some tests is brief. When possible, all of the information available has been included in tables. In some cases the tests are treated individually, because of lack of uniformity in the information and the difficulty of including in the tables all the data available.

TESTS IN BEEHIVE OVENS.

OVENS AT SESSER, ILL.

Two experimental beehive ovens were built by a coal company at Sesser. The ovens were both 12 feet in diameter, one of them being 6 feet 3 inches high and the other 7 feet 3 inches high. Several tests were made of lump and run-of-mine coal, and of screenings from the company's mine, district 6, seam 6, including tests of charges of uncrushed and of finely crushed coal. The weights varied from 5 to 6½ tons and the time of coking from 34 to 82 hours.

As a result of the tests the opinion was stated that coke of good quality, suited for any commercial use, could be made from this coal. Washing was necessary for the run-of-mine coal and the screenings, and was recommended for the lump coal. Fine crushing was necessary in order to decrease the effect of unfused material in coke made from unwashed coal. The yield of coke was about 57 per cent. The same coal was tested by the United States Geological Survey, the results being given in Table 4.^a It has also been tested in by-product ovens.

COKING TESTS MADE BY UNITED STATES GEOLOGICAL SURVEY.

Before the establishment of the Bureau of Mines the United States Geological Survey made coking tests in beehive ovens of coals from different parts of the country. From 1904 to 1907 the work was carried on at St. Louis, Mo. In 1907 the equipment was transferred to Denver, Colo., where work was continued until the summer of 1909. The testing of fuels was subsequently transferred to the Bureau of Mines, and the results of the work have been published in bulletins of the Geological Survey^b and the Bureau of Mines.^c

^a See results for Denver Nos. 15A and 15B (p. 34).

^b Stammer, F. W., Coking tests: U. S. Geol. Survey Prof. Paper 48, pt. 3, 1906, p. 1326; Belden, A. W., Coking tests: U. S. Geol. Survey Bull. 290, 1906, p. 38; Belden, A. W., Coking tests: U. S. Geol. Survey Bull. 332, 1908, p. 32; Moldenke, R., Belden, A. W., Delamater, G. R., Washing and coking tests of coal and cupola tests of coke: U. S. Geol. Survey Bull. 336, 1908, p. 18.

^c Belden, A. W., Delamater, G. R., Groves, J. W., and Way, K. M., Washing and coking tests of coal, at Denver, Colo., July 1, 1908, to June 30, 1909: Bull. 5, Bureau of Mines, 1910, 62 pp., 1 fig.

During this time 45 coking tests were made of 20 different Illinois coals. The tests at St. Louis were made in a battery of three beehive ovens of standard size and shape. Most of the tests were made in the two end ovens. After the first few tests the bottom of one of the end ovens was raised 8 inches in order to bring the charge nearer the dome and effect a more rapid penetration of heat. The tests at Denver were made in a battery of two beehive ovens, 12 feet in diameter, one being 7 feet high, the other 6 feet 3 inches.

The coal charged to the ovens was passed through rolls which reduced it to $1\frac{1}{2}$ inches or smaller in the tests of Illinois No. 1 to Illinois No. 5 coals; in the other tests, unless otherwise stated, the coal was crushed so that practically all of it passed through a 10-mesh sieve. The weight of charges and the coking time were varied with the same coal and the effect of these changes upon the quality of the coke was noted.

A brief summary of these tests is given in Table 4. More detailed information is given in the bulletins cited. The designation of the coals is that used by the United States Geological Survey and the Bureau of Mines.

The analyses of coals are on the basis of coal as charged. Moisture in the washed coal is often higher than in the coal "as received," some water being retained from the washing process.

The analyses of coke are on the "as-received" basis, and the results include moisture from water used in quenching.

The yield of coke is based on the weight of coal as charged to the oven, so that owing to the high moisture content in some of the coals, the percentage yield of coke calculated in this way is low.

The item "coke" includes everything that remains after thorough shaking on a fork having tines $1\frac{1}{2}$ inches apart.

The item "breeze" includes everything that will pass through a fork having tines $1\frac{1}{2}$ inches apart. Its percentage is higher than from regular operations.

Illinois No. 16.	Herrin, Williamson County, Ill., district 6, seam 6.	Lump and egg.	Coal.	9.79	30.35	51.79	8.07	1.09	.008	55.5	9.1	64.9	.87	1.56	47	53	53.1	86.2	Dull-gray color; poor coke physically.
Illinois No. 19-A.	Zoeller, Franklin County, Ill., district 6, seam 6.	1-inch.	Coal.	12.76	27.24	51.73	8.24	.49											No coke produced.
Illinois No. 19-B.	do.	3-inch.	Coal.	9.64	30.66	50.22	9.46	.53											Do.
Illinois No. 20.	Stanton, Macoupin County, Ill., district 7, seam 6.	Screenings.	Coal.	17.04	32.59	40.77	9.60	3.23											Dull gray color; dense, gross fracture bed.
Illinois No. 21.	Troy, Madison County, Ill., district 7, seam 6.	Lump.	Coal.	14.36	34.61	42.63	8.40	3.22											Dull gray color; a little better than from raw coal.
Illinois No. 22-B.	Maryville, Madison County, Ill., district 7, seam 6.	Through 2 inch screen.	Coal.	13.37	31.07	43.15	12.31	1.46											Slightly fused coal; no cell structure.
Illinois No. 23-A.	Donkville, Madison County, Ill., district 7, seam 6.	Lump.	Coal.	17.45	30.01	44.74	7.80	1.10											Slightly fused coal; no coke.
Illinois No. 23-B.	do.	Slack.	Coal.	11.98	33.87	37.73	16.43	.74											Poor coke; dull color; irregular breakage.
Illinois No. 24-A.	New Baden, Clinton County, Ill., district 7, seam 6.	Screenings.	Coal.	15.85	35.02	40.57	8.56	3.27											Poor coke; heavy; black ends.
Illinois No. 24-B.	do.	Lump.	Coal.	15.83	35.08	40.16	8.03	3.25											Good heavy coke; ash and sulphur high.
Illinois No. 25.	Germantown, Clinton County, Ill., district 7, seam 6.	Run-of-mine and lump.	Coal.	13.28	30.98	39.03	17.76	.05											Do.
Illinois No. 26.	Linton, Logan County, Ill., district 4, seam 6.	Run-of-mine.	Coal.	15.18	32.13	43.46	9.23	3.07											Not as good as from No. 25-A coal.
Illinois No. 27.	Auburn, Sangamon County, Ill., district 7, seam 6.	do.	Coal.	8.93	35.22	46.29	9.56	3.41											No coke.
Illinois No. 28.	Herrin, Williamson County, Ill., district 6, seam 6.	Lump.	Coal.	12.96	33.01	38.60	14.43	.09											Soft dense coke.
Illinois No. 29.	do.	Screenings.	Coal.	13.40	33.83	43.66	9.11	2.99											Do.
Illinois No. 30.	do.	Lump.	Coal.	15.18	33.46	41.53	9.83	2.73											Soft coke; poor cell structure.
Illinois No. 31.	do.	Screenings.	Coal.	3.27	3.35	75.98	15.40	2.50											Poor coke.
Illinois No. 32.	do.	Lump.	Coal.	16.38	34.26	41.57	7.73	2.28											
Illinois No. 33.	do.	Screenings.	Coal.	3.51	1.55	81.14	13.80	3.40											
Illinois No. 34.	do.	Lump.	Coal.	9.37	30.38	53.36	6.99	1.09											
Illinois No. 35.	do.	Screenings.	Coal.	2.83	.00	86.12	10.46	.98											

* Charged as run-of-mine coal.

TESTS OF FRANKLIN COUNTY COAL AT REPUBLIC, PA.

The following information is taken from the report of a test of coal from Franklin County, district 6, seam 6, made by a large iron and steel company. The coal was coked in the company's beehive ovens at Republic, Pa., and the coke was tested in the company's blast furnace at Sharon, Pa.

COKING TESTS.

Ten single-row ovens were set aside for the test, the ovens being in good condition at the time of starting. First, the ovens were charged with run-of-mine coal. The coke resulting was practically worthless for blast-furnace use. Next, the ovens were charged with crushed coal and burned 72 hours. No great difference could be noticed in the coke so made. The ovens were then charged with crushed coal, wet, as though it had gone through a washer. The wetting appeared to tamp the coal in the oven and a heavier and denser coke was produced, the cells being much smaller than those in the coke made from dry coal.

A summary of the results of the coking tests follows:

Summary of results of coking tests of Illinois coal at Republic, Pa.

Total amount of coal charged, tons.....	438. 19
Total amount of coke made, tons.....	224. 30
Duration of test, days.....	16
Coal charged per day, tons.....	27. 38
Coke made per day, tons.....	15. 26
Tons of coal to net tons of coke.....	1. 90
Total ovens drawn.....	70
Coal charged per oven, tons.....	6. 25
Coke made per oven, tons.....	3. 20
Yield of commercial coke, per cent.....	51. 19
Yield of coke and refuse, per cent.....	57. 25
Average temperature of ovens, °F.....	2, 546

Average results of analyses of coal and coke.

	Coal.	Coke.
Volatile matter, per cent.....		2. 84
Fixed carbon, per cent.....		83. 62
Ash, per cent.....	7. 31	13. 54
Sulphur, per cent.....	1. 26	1. 15
Phosphorus, per cent.....	. 008	. 0115

CONCLUSIONS.

The following conclusions, based on the results of the tests, were drawn:

The coal would not make satisfactory coke without crushing.

The best coke from the test was obtained from coal crushed so that it would pass through a screen in which the holes were not greater than $\frac{3}{16}$ inch in diameter.

A heavier and denser coke was made by wetting the coal.

The best way to coke this coal was to start the ovens with heavy draft, gradually closing the draft as the oven was burned off. Driving the volatile matter off quickly gave a harder and better colored coke.

The average tonnage per oven, 12 feet 6 inches diameter, was not over 4 tons of coke.

The yield of coal to coke was 10 per cent less than the average yield obtained from Connellsville coal.

The coke could probably have been improved if the coal had been washed, so as to reduce the ash and sulphur contents.

USE OF COKE IN BLAST FURNACE.

The coke was found to be porous and of low specific gravity, weighing about 14.2 per cent less than the coke generally used. Although it did not look soft and had a fair structure, it acted in the furnace as if it did not have much strength, and when it was unloaded and handled, it broke into small blocks. The Illinois coke was started in the furnace on the twentieth round, day turn, December 16, and its use continued until the first round of the night turn of December 17, or 159 rounds. At the same time a change was made in the lime charged, to allow for the difference in composition of this coke. After the coke had been used about four hours the furnace began to slip, became somewhat cold, and worked in this manner until all the Illinois coke had been used.

COKING TESTS AT LINTON, IND.

In about 1908 four coking tests were made by a coal and coke company at Linton, Ind., of different sizes of coal from Franklin County, district 6, seam 6. They were made in a battery of 10 beehive ovens, which were 12 feet in diameter and 8 feet high. There were no scales at the plant and the yields given are calculated on railroad weights. The sampling and analyzing were done carefully and as accurately as was possible.

All the coal coked well and ignited quickly in the ovens. The coke from the screenings showed that they should be washed to improve the quality. The average yield of coke from the coal as charged was about 52 per cent. Breeze, including ash, was 6.7 per cent. Data regarding the tests are given in Table 5 following:

TABLE 5.—Results of coking tests of Franklin County coal at Linton, Ind.

Test. No.	Size of coal.	Number of ovens charged.	Weight of coal charged.	Weight of coke produced.	Yield of coke.	Time of burning.	Time of ignition.	Results of analysis.						
								Kind of sample.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.
			Lbs.	Lbs.	P. ct.	Hrs.	Mins.		P. ct.	P. ct.	P. ct.	P. ct.	P. ct.	P. ct.
1	Run of mine...	10	90,000	46,100	51.2	45.5	27	Coal...	1.10	37.27	51.50	10.13	1.55	
2	2-inch screen- ings.	9	81,000	42,100	52	45	10	Coke...	1.37	.76	82.55	15.02	1.13	0.082
3	Egg.....	10				60	4.5	Coal...	8.06	33.16	47.80	10.96	1.16	.014
4	do.....	9	242,200	127,900	52.4	72	17	Coke...	1.87	2.62	79.42	16.69	1.23	.013
								Coal...	7.65	34.80	48.48	9.07	1.25	.007
								Coke...	1.76	.37	83.14	14.74	.90	.010
								Coal...	7.50	34.24	48.12	10.14	.94	.011
								Coke...	1.94	1.01	81.22	15.83	1.12	.021

TESTS IN BY-PRODUCT OVENS.

Most of the tests in by-product ovens were made on a large scale under actual operating conditions. They were carried on during regular operation by charging one or more ovens with test coal. A few "box" tests were made, as follows: About 500 pounds of coal were put in a suitable container and placed in an oven. The yield and the quality of coke produced could be determined, but in most cases it was not possible to determine the yield of by-products. These passed into the regular recovery system and mixed with the products of distillation from the other ovens in the battery.

In most of the tests the by-products were determined in the laboratory. Laboratory tests usually show higher results than those obtained in commercial practice, and the results should not be compared directly with oven yields unless the method has been correlated with practice; even then it may be necessary to apply a correction factor. Conditions in the laboratory are more uniform and can be controlled better than in the ovens; therefore the laboratory test can be used as a guide to determine whether the ovens are operating satisfactorily and whether the maximum yield of by-products is being obtained. In some plants experimental ovens are being equipped with separate recovery systems in which by-products from one or more ovens can be treated separately and the yields determined. This arrangement will allow experiments to be made in a more satisfactory manner and should result in making available much valuable information.

DETAILED RESULTS OF TESTS.

Detailed results of coking tests in by-products ovens follow.

Regarding the results for tests 15, 16, 18, 26, 27, 28, 29, 32, 33, and 34, the figures for yields of by-products are from oven tests. These were made at Chattanooga, Tenn., in a small battery of Rob-

erts ovens, the entire battery being operated on the test coal. The data were furnished by Mr. Arthur Roberts, president of the American Coal & By-products Coke Co. No detailed description of the coke from these tests was obtained, but a representative of the company stated that, in his opinion, it was suitable for metallurgical use although it had not been actually tried in a furnace. It is understood that a battery of Roberts ovens which will use Illinois coal for making metallurgical coke is to be built near St. Louis.

TEST 1.

In test 1, about 300 tons of $\frac{1}{4}$ -inch washed coal from Williamson County, district 6, seam 6, was used and the resultant coke was tried as locomotive fuel by the Illinois Central Railroad Co. The result is not known.

The coke looked rough and had poor cell structure, but was comparatively strong and shipped fairly well. The coke was high in ash and suitable for fuel and domestic purposes only. It tended to break in cross fractures and in the shatter test an average of 53.6 per cent stayed on the screen. The shatter test^a is used to determine the relative breakage during handling. The results given are the percentage remaining on a 2-inch screen.

Tabulated results of the test follows:

Data regarding test 1.

Wet coal charged, pounds.....	611, 160
Dry coal charged, pounds.....	531, 458
Wet coke produced, pounds.....	400, 000
Dry coke produced, pounds.....	390, 393
Average coking time, hours.....	23 $\frac{1}{4}$

Analysis of dry coal and coke from test 1.

	Coal.	Coke.
Volatile matter, per cent.....	32.7	1.5
Fixed carbon, per cent.....	56.8	84.5
Ash, per cent.....	11.0	14.0
Sulphur, per cent.....	.84	.72
Phosphorus, per cent.....	.009	.011

TEST 2.

In test 2, one car of washed coal from Franklin County, district 6, seam 6, was used. It was pulverized before being charged, so that 70.4 per cent passed through $\frac{1}{8}$ -inch openings and 94.4 per cent through $\frac{1}{4}$ -inch openings. Three charges were made in a single oven. In one the coal was charged loose in the ordinary manner and in the other two it was compressed before being charged. The coking time was longer, about 46 hours, and the temperature of the ovens lower than usual, being approximately 900° to 950° C.

^a For a description of the apparatus and method used in conducting the test, see Belden, A. W., Metallurgical coke: Tech. Paper 50, Bureau of Mines, 1913, p. 28.

The coke from the loose charge was of fairly good size with no indication of sponge. It shattered badly when dropped, breaking into nut size with a large amount of breeze. The cell structure was fairly regular with a rather large percentage of cell space but with thin cell walls. The fusion was complete. The slower coking slightly reduced the formation of fingers but did not add much to the thickness of the cell walls. The coke from the compressed charges was similar to that from the coal charged loose, there being little change in toughness or resistance to crushing. It was not of the highest metallurgical grade. The yields of coke, gas, and tar were not determined. The yield of ammonium sulphate by the laboratory method was 34.7 pounds per ton of coal.

Analysis of coal and coke from test 2.

Constituent.	Loose charge.		Compressed charges.			
	Coal.	Coke.	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	11.11	2.12	18.60	1.72	14.64	1.80
Volatile matter, per cent.....	37.75	1.45	38.85	1.45	38.84	1.80
Fixed carbon, per cent.....	54.75	87.75	54.15	87.91	55.44	88.14
Ash, per cent.....	7.50	10.80	7.00	10.64	7.72	10.56
Sulphur, per cent.....	1.02	.92	1.09	.96	.96	1.01
Phosphorus, per cent.....	.008	.009	.010	.014	.005	.013

TEST 3.

In test 3, a car of coal from Franklin County, district 6, seam 6, was used. It was crushed so that 89.6 per cent passed through $\frac{1}{8}$ -inch openings and 96.4 per cent through $\frac{1}{4}$ -inch openings. Five single-oven charges were made under different conditions.

First, Illinois coal only was charged into a very hot oven and coked in 18 hours. The coke varied in size from very small up to pieces 6 inches across the butt end, with a tendency to break into needlelike pieces. It showed considerable cross fracturing and when dropped it broke into small pieces with a large proportion of breeze. The cell structure was fairly uniform throughout the length of the pieces, the walls were thin, and the cell space was 42.5 per cent. The top of the charge had a tendency to sponge and a few pieces in the charge had a granular structure with no cells. At the center of the charge the coke had a tendency to curl. Because of its friability and weakness it would not be considered a high-grade metallurgical fuel.

Second, Illinois coal only was charged into an oven of moderate heat and coked in 25 hours. The coke appeared much the same as from the first charge except that it tended to form in more blocky pieces.

Third, Illinois coal only was compressed in a stamping machine before being charged into a moderately hot oven, and was coked in

24 hours. The coke appeared much the same as from the uncompressed charge, except that it was heavier and denser with a smaller percentage of cell space. There was little or no improvement in friability and strength of cell walls. Its use would be limited to that of the coke from uncompressed coal.

Fourth, 85 per cent of Illinois coal was mixed with 15 per cent low-volatile coking coal, charged into a hot oven, and coked in 18 hours. The coke was in pieces of irregular size up to 4 inches across the butt. It showed no tendency to break into fingerlike pieces, but a slight tendency to cross check. When dropped it had an irregular fracture, breaking into small pieces with considerable breeze. It had a poor external appearance with a dead-black surface and a dull lustre. The cell structure was rather open and irregular, the walls being fairly thick. A few pieces in the charge had a granular structure. This coke was somewhat stronger and better than from Illinois coal alone.

Fifth, a mixture of 75 per cent of Illinois coal and 25 per cent low-volatile coal was charged into a hot oven and coked in 18 hours. The coke was in pieces of irregular size up to 8 inches across the end. In general appearance it was about the same as that from the 85-15 per cent mixture. It was the best coke of any produced in the test.

The yields of coke, gas, and tar were not determined. The yield of ammonium sulphate by the laboratory method was 29.68 per cent per ton of coal.

Analysis of coal and coke and physical properties of coke, test 5.

Constituent.	Illinois coal only.				Illinois coal only, compressed.		85 per cent Illinois coal and 15 per cent low-volatile coal.		75 per cent Illinois coal and 25 per cent low-volatile coal.	
	Coal.	Coke.	Coal.	Coke.	Coal.	Coke.	Coal.	Coke.	Coal.	Coke.
Moisture, per cent...	9.01	1.66	8.00	1.22	17.09	4.00	12.82	1.46	9.01	1.46
Volatile matter, per cent.....	35.94	.97	36.47	1.11	35.95	1.23	32.98	.85	30.82	.85
Fixed carbon, per cent.....	54.91	86.66	55.33	83.79	52.80	83.47	56.66	86.29	59.83	86.29
Ash, per cent.....	9.15	12.37	8.20	15.10	11.25	15.30	10.36	12.86	9.35	12.86
Sulphur, per cent...	1.08	.87	1.08	.89	1.15	.96	1.10	.81	1.01	.81
Phosphorus, per cent.	.006	.007	.006	.007	.006	.008	.007	.009	.005	.009
Apparent specific gravity...		0.81				0.85				
Real specific gravity.....		1.47				1.35		1.35		
Shatter test—per cent retained on 2-inch screen....		50		47		45.0		69.7		

TEST 4.

For test 4, seven cars of coal from Franklin County, district 6, seam 6, were used. The coal was crushed so that 79.5 per cent passed through $\frac{1}{8}$ -inch openings and 96.1 per cent through $\frac{1}{4}$ -inch openings. Twenty-eight ovens were charged, and the average coking time was 23 $\frac{1}{2}$ hours.

The general appearance of the coke is shown in Plate III, A. A characteristic feature was the tendency to form curved pieces, especially at the bottom of the oven. The coke had a marked tendency to break into fingerlike pieces about 1 inch in diameter. It was remarkably free from cross fractures but was very brittle. When tumbled in a pile it gave a rustling sound. The cell structure was fairly uniform, the cells being small and numerous, and the walls thin. Many pockets were found where the coke did not seem to fuse.

Approximately 290 tons of this coke was used in a copper blast furnace to determine its suitability for such use. The fact was demonstrated that the furnace could be run on this coke.

Data from test 4.

Coal charged, tons.....	379
Yield of dry coke, including breeze, per cent.	76.9
(NH ₄) ₂ SO ₄ per ton, from laboratory test, pounds.....	29.36
Shatter test, per cent retained on 2-inch screen.....	37.2
Apparent specific gravity of coke.....	0.85
Real specific gravity of coke.....	1.43
Porosity of coke, per cent.....	42

Analysis of coal and coke, test 4.

	Coal.	Coke.
Moisture, per cent.....	8.28	9.05
Volatile matter, per cent.....	37.09	.87
Fixed carbon, per cent.....	52.71	84.85
Ash, per cent.....	10.20	14.28
Sulphur, per cent.....	1.25	1.02
Phosphorus, per cent.....	.008	.015

TEST 5.

In test 5, coal from Franklin County, district 6, seam 6, was used. The coal was crushed so that 92.3 per cent passed through $\frac{1}{8}$ -inch openings. Four ovens were charged, three of them under regular operating conditions. In the fourth the gas was cut down in order that the effect of low heat might be observed. In the first three ovens the average coking time was 18 $\frac{1}{2}$ hours, and in the fourth the coke was pushed green in 19 $\frac{1}{2}$ hours.

In addition, three lots of coal, weighing about 600 pounds each, were specially prepared in boxes before being inserted in the ovens. Box 1 was filled with wet compressed coal, which was pushed in

20 hours after charging; it was rather underdone. Box 2 was filled with dry compressed coal, which was put in an oven at the ordinary temperature, and pushed in 18½ hours. Box 3 was filled with wet uncompressed coal, which was coked at the same temperature as the dry coal in box 2.

The coke from the first three ovens was of medium size and showed few cross fractures. It was fingerlike and brittle and showed partly fused material. The cells were small and somewhat irregular. The coke from the fourth oven, in which the charge was coked at low temperature and pushed green, showed a more blocky structure than the others, but it was soft and easily broken. The box tests seemed to indicate that wetting and compressing the coal improved the quality of the coke, making it tougher and more blocky and giving a larger cell structure. Coke from the wet uncompressed coal was blocky and had a large cell structure. The dry compressed coal gave a coke similar to that from the wet compressed coal.

Data from four oven charges, test 5.

Dry coal charged, pounds.....	38,996
Dry coke produced, pounds.....	29,947
Coke, per cent.....	76.8
Result of shatter test, per cent held on 2-inch screen.....	51.6
Apparent specific gravity of coke.....	.78
Real specific gravity of coke.....	1.62
Porosity, per cent.....	49.6
(NH ₄) ₂ SO ₄ per ton, from laboratory test, pounds.....	27
Tar per ton, gallons.....	6
Gas per ton, cubic feet.....	9,700

Analysis of coal and coke, test 5.

	Coal.	Coke.
Moisture, per cent.....	8.45	11.40
Volatile matter, per cent.....	35.21	1.81
Fixed carbon, per cent.....	54.30	84.25
Ash, per cent.....	10.49	13.94
Sulphur, per cent.....	1.10	.82
Phosphorus, per cent.....	.011	.013

• TEST 6.

In test 6, washed slack from Franklin County, district 6, seam No. 1 was used. The coal was crushed so that 72 per cent passed through ½-inch openings and 84 per cent through ¼-inch openings. One oven charge was made of Illinois coal alone, and box tests were made of mixtures of 90 per cent Illinois coal and 10 per cent low-volatile coal, 80 per cent Illinois coal and 20 per cent low-volatile coal, 70 per cent Illinois coal and 30 per cent low-volatile coal, and 60 per cent Illinois coal and 40 per cent low-volatile coal. The coking time in each case was 16 hours.



A. COKE FROM 100 PER CENT ILLINOIS (FRANKLIN COUNTY, DISTRICT 6, BED 6) COAL, TEST 4. FEW CROSS FRACTURES; TENDENCY TO FINGER AND FORM CURVED PIECES.



B. COKE FROM 100 PER CENT ILLINOIS (DISTRICT 6, BED 6) COAL, TEST 9. FAIRLY OPEN CELL STRUCTURE; TENDENCY TO BREAK INTO FINGERLIKE PIECES.

The coke from the oven charge was fingerlike and brittle, with few cross checks. The coke from the box tests was more blocky, with irregular cell structure.

Analysis of coals used in test 6.

	Illinois coal.	Low-volatile coal.
Moisture, per cent.....	0.50	0.30
Volatile matter, per cent.....	34.03	18.73
Fixed carbon, per cent.....	55.52	77.31
Ash, per cent.....	9.95	3.66
Sulphur, per cent.....	1.22	.76

Analysis of coke, test 6.

Constituent.	Illinois coal only.	90 per cent Illinois coal and 10 per cent low-volatile coal.	80 per cent Illinois coal and 20 per cent low-volatile coal.	70 per cent Illinois coal and 30 per cent low-volatile coal.	60 per cent Illinois coal and 40 per cent low-volatile coal.
Volatile matter, per cent.....	2.23	2.95	2.93	2.08	1.54
Fixed carbon, per cent.....	87.50	86.57	88.01	89.98	91.04
Ash, per cent.....	10.18	10.48	9.06	7.94	7.42
Sulphur, per cent.....	1.04	0.86	.86	.93	.90

TEST 7.

Coal from Williamson County, district 6, seam 6, was used in test 7. Oven charges were made of mixtures of 70 per cent Illinois coal with 30 per cent low-volatile coal, 60 per cent Illinois coal with 40 per cent low-volatile coal, and 50 per cent Illinois coal with 50 per cent low-volatile coal. The coals were pulverized and mixed before being charged.

The mixtures required a somewhat longer coking time than the regular charges used in the ovens. The coke from the 70-30 mixture was very poor, that from the 60-40 mixture was somewhat better but was slatey and broke considerably on handling. The coke from the 50-50 mixture was better than that from either of the other charges; it was tougher and had less tendency to cross check.

Analysis of coal and coke, test 7.

Constituent.	70 per cent Illinois coal and 30 per cent low-volatile coal.		60 per cent Illinois coal and 40 per cent low-volatile coal.		50 per cent Illinois coal and 50 per cent low-volatile coal.	
	Coal.	Coke.	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	1.00	0.10	2.15	0.04	2.12	0.17
Volatile matter, per cent.....	25.45	.85	25.99	3.71	28.56	1.36
Fixed carbon, per cent.....	67.26	87.45	64.85	86.23	61.17	88.29
Ash, per cent.....	6.29	11.60	7.21	10.02	8.15	10.18
Sulphur, per cent.....	.98	1.07	.98	.92	1.18	.89

TEST 8.

Coal from district 6, seam 6, was used in test 8. Oven charges were made with Illinois coal only and with a mixture of 20 per cent Illinois coal and 80 per cent low-volatile coal. The yields of products when Illinois coal only was used were determined by a laboratory test.

The coke from the charge of Illinois coal alone was in piece of fair size, with rather open cell structure. It had a porosity of 48.2 per cent, and in a shatter test 71.3 per cent was held on a 2-inch screen. Unfused material was quite noticeable.

Plate IV shows coke from the mixture. The somewhat irregular cell structure, the cross fractures, and the nonfused material are clearly shown. The coke had a porosity of 48.2 per cent, and in a shatter test 80.4 per cent was held on a 2-inch screen.

Yields per ton of 100 per cent Illinois coal as determined in laboratory test 8.

Coke, per cent.....	69.6
Tar, gallons.....	7.4
(NH ₄) ₂ SO ₄ per ton, pounds.....	33.4
Light oil, pounds.....	19.6
Gas (including CO ₂ and H ₂ S), cubic feet at 15° C. and 30 inches Hg.....	10,312

Analysis of coal and coke produced in ovens, test 8.

Constituent.	Illinois coal alone.		20 per cent Illinois coal and 80 per cent low-volatile
	Coal.	Coke.	Coal.
Moisture, per cent.....	2.75		3.32
Volatile matter, per cent.....	39.41	3.32	20.04
Fixed carbon, per cent.....	51.53	84.16	73.20
Ash, per cent.....	9.06	12.52	6.76
Sulphur, per cent.....	2.03	1.65	1.02
Phosphorus, per cent.....	.023	.031	.008

TEST 9.

In test 9 coal from district 6, seam 6, was used. Oven charges were made with Illinois coal only and with a mixture of 20 per cent Illinois coal and 80 per cent low-volatile coal. By-products from both charges were determined by a laboratory test.

The coke from Illinois coal alone is shown in Plate III, B. The tendency to break up into small fingerlike pieces is noticeable. The cell structure was fairly open and there was some spongy material. The coke had a porosity of 49.9 per cent, and in a shatter test 71.3 per cent was retained on a 2-inch screen.

Coke from the mixture is shown in Plate V. The pieces were of fair size with uniform cell structure. There was some tendency to break in cross fractures, but the coke stood handling well. It had a porosity of 48.2 per cent.



COKE FROM 20 PER CENT ILLINOIS (DISTRICT 6, BED 6) COAL AND 80 PER CENT LOW-VOLATILE COAL, TEST 8. SLIGHTLY IRREGULAR CELL STRUCTURE, NONFUSED MATERIAL, TENDENCY TO CROSS FRACTURE.



of 47.5 per cent, and in a shatter test 80.5 per cent was retained on a 2-inch screen.

Yields per ton of coal as determined in laboratory test 9.

	Illinois coal alone.	20 per cent Illinois coal and 80 per cent low-volatile coal.
Coke, per cent.....	70.2	81.3
Tar, gallons.....	7.2	4.0
(NH ₄) ₂ SO ₄ per ton, pounds.....	35.4	20.8
Light oil, pounds.....	19.4	14.2
Gas (including CO ₂ and H ₂ S), cubic feet at 15° C. and 30 inches Hg.....	10,396	10,088

Analysis of coal and coke produced in ovens, test 9.

Constituent.	Illinois coal alone.		20 per cent Illinois coal and 80 per cent low-volatile coal.	
	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	7.00		4.40	
Volatile matter, per cent.....	28.83	1.40	22.26	1.21
Fixed carbon, per cent.....	60.85	82.78	69.10	87.79
Ash, per cent.....	10.32	15.82	8.64	11.00
Sulphur, per cent.....	1.59	1.40	.83	.83
Phosphorus, per cent.....	.005	.007	.006	.006

TEST 10.

Coal from district 6, seam 6, was used in test 10. Oven charges were made with Illinois coal alone and with a mixture of 20 per cent Illinois coal and 80 per cent low-volatile coal. The by-products from the charge of Illinois coal alone were determined by laboratory test.

Plate VI shows coke from the Illinois coal alone. The tendency to form fingers and to curl are noticeable. The coke was rather soft, had an irregular cell structure, and contained nonfused material. It had a porosity of 50.6 per cent, and in a shatter test 74 per cent was retained on the screen.

Coke from the mixture is shown in Plate I, B (p. 16). It was in blocky pieces, the tendency to finger being reduced by the addition of the low-volatile coal. It had a good color and contained little unfused material. The porosity was decreased to 48.4 per cent, and in the shatter test the proportion retained on the screen was raised to 81.5 per cent.

Yields per ton of Illinois coal alone, as determined in laboratory test 10.

Coke, per cent.....	70.5
Tar, gallons.....	6.0
(NH ₄) ₂ SO ₄ per ton, pounds.....	37.4
Light oil, pounds.....	19.8
Gas (including CO ₂ and H ₂ S), cubic feet at 15° C. and 30 inches Hg.....	10,663

Analysis of coal and coke produced in ovens, test 10.

Constituent.	100 per cent Illinois coal.		20 per cent Illinois coal 80 per cent volatile	
	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	6.47		3.20	
Volatile matter, per cent.....	31.47	2.17	20.51	
Fixed carbon, per cent.....	55.09	83.56	72.31	
Ash, per cent.....	11.84	18.27	7.18	
Sulphur, per cent.....	1.00	.84	.70	
Phosphorus, per cent.....	.014	.019	.007	

TEST 11.

In test 11, coal from district 5, seam 5, was used. Oven charges were made with Illinois coal only and with a mixture of 20 per cent Illinois coal and 80 per cent of low-volatile coal. By-products from the Illinois coal alone were determined by laboratory test.

Coke from the Illinois coal alone had a tendency to form fingers and contained some sponge. There was little nonfused material. Porosity was 44.3 per cent, and in the shatter test 74.1 per cent was retained on the screen.

The mixture produced a hard coke with a medium open but uniform cell structure. It was less friable than that from the Illinois coal alone, and in the shatter test 83.9 per cent was retained on screen. The porosity was 41.6 per cent.

Yields per ton of Illinois coal alone, as determined in laboratory test 11.

Coke, per cent.....	73.5
Tar, gallons.....	4.4
(NH ₄) ₂ SO ₄ per ton, pounds.....	34.2
Light oil, pounds.....	21.6
Gas (including CO ₂ and H ₂ S), cubic feet at 15° C. and 30 inches Hg.....	10,702

Analysis of coal and coke produced in ovens, test 11.

Constituent.	100 per cent Illinois coal.		20 per cent Illinois coal 80 per cent volatile	
	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	4.20		3.13	
Volatile matter, per cent.....	33.80	2.97	21.80	
Fixed carbon, per cent.....	55.36	83.39	71.08	
Ash, per cent.....	9.84	13.64	7.12	
Sulphur, per cent.....	2.62	2.26	1.10	
Phosphorus, per cent.....	.011	.009	.010	



COKE FROM 100 PER CENT ILLINOIS (DISTRICT 6, BED 6) COAL, TEST 10. TENDENCY TO FINGER, IRREGULAR CELL STRUCTURE, NONFUSED MATERIAL.



TEST 12.

In test 12, coal from district 7, seam 6, was used. Oven tests were made with Illinois coal alone and with a mixture of 20 per cent Illinois coal and 80 per cent low-volatile coal. A laboratory by-product test was made with the Illinois coal alone.

Plate VII shows coke from the test in which Illinois coal was used alone. It was soft and had an irregular cell structure. A large amount of nonfused material was noticeable. The porosity was 51.7 per cent, and in the shatter test 75.1 per cent was retained on the screen.

The general appearance of coke from the mixture is shown in Plate VIII. Fusion was fairly complete but the cell structure was irregular and the coke had a tendency to cross fracture. The porosity was 49.4 per cent, and in the shatter test 81.0 per cent was retained on the screen.

Yields per ton of Illinois coal alone, as determined in laboratory test 12.

Coke, per cent.....	72.5
Tar, gallons.....	6.2
(NH ₄) ₂ SO ₄ , per ton, pounds.....	32
Light oil, pounds.....	17.8
Gas (including CO ₂ and H ₂ S), cubic feet at 15°C. and 30 inches Hg.	9,747

Analysis of coal and coke produced in ovens, test 12.

Constituent.	Illinois coal alone.		20 per cent Illinois coal and 80 per cent low-volatile coal.	
	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	3.13		2.73	
Volatile matter, per cent.....	35.39	1.60	20.25	1.27
Fixed carbon, per cent.....	52.07	82.44	71.67	89.13
Ash, per cent.....	12.45	15.96	8.08	9.60
Sulphur, per cent.....	.86	.93	.70	.74
Phosphorus, per cent.....	.014	.017	.008	.012

TEST 13.

Coal from district 6, seam 6, was used in test 13. Oven tests were made with Illinois coal alone and with a mixture of 20 per cent Illinois coal and 80 per cent low-volatile coal. A laboratory by-product test was made with Illinois coal alone.

Coke from the Illinois coal alone is shown in Plate IX. The cell structure was fairly uniform and there were few cross fractures. The coke was in small pieces and had a tendency to finger. The porosity was 45.4 per cent, and in the shatter test 61.1 per cent was retained on the screen.

Coke from the mixture is shown in Plate X. The fusion was and the cell structure fairly uniform. The size of pieces was increased, and the friability was lessened by the addition of volatile coal. The porosity was raised to 49.8, and in the shatter test the proportion retained on the screen was increased to 89.9 per cent.

Yields per ton of Illinois coal alone, as determined in laboratory test 13.

Coke, per cent.....	71.5
Tar, gallons.....	6
(NH ₄) ₂ SO ₄ per ton, pounds.....	32
Light oil, pounds.....	24.5
Gas (including CO ₂ and H ₂ S), cubic feet at 15°C. and 30 inches Hg.	10, 912

Analysis of coal and coke produced in ovens, test 13.

Constituent.	Illinois coal alone.		20 per cent Illinois coal and 80 per cent low-volatile coal.
	Coal.	Coke.	Coal.
Moisture, per cent.....	2.90		2.20
Volatile matter, per cent.....	35.13	2.05	20.97
Fixed carbon, per cent.....	64.29	84.71	71.72
Ash, per cent.....	10.58	13.24	7.49
Sulphur, per cent.....	1.58	1.32	.90
Phosphorus, per cent.....	.015	.019	.007

TEST 14.

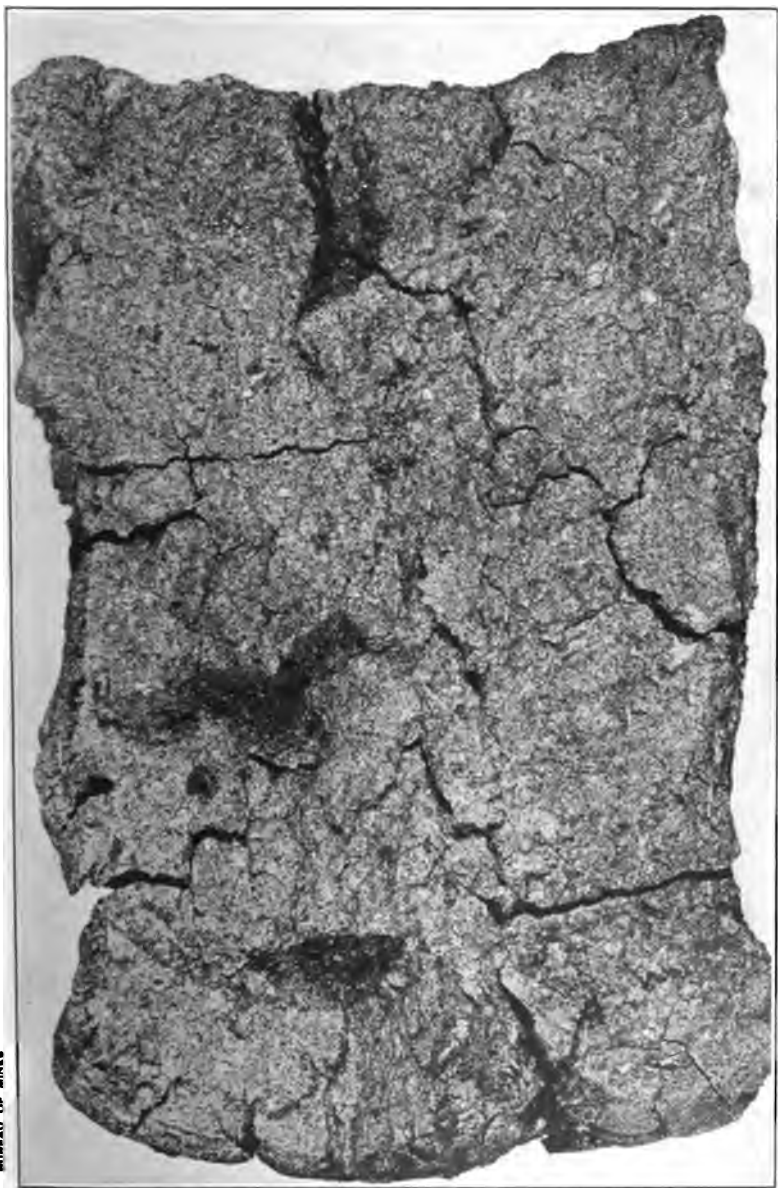
In test 14, coal from district 6, seam 6, was used. Oven charges were made with Illinois coal only and with a mixture of 20 per cent Illinois coal and 80 per cent low-volatile coal. The by-products from both charges were determined in the laboratory.

The general appearance of the coke from the Illinois coal is shown in Plate XI. The fusion was fairly good but the cell structure was somewhat irregular. It contained considerable non-fusible material. The porosity was 47.6 per cent, and in the shatter test 79.7 per cent was retained on the screen.

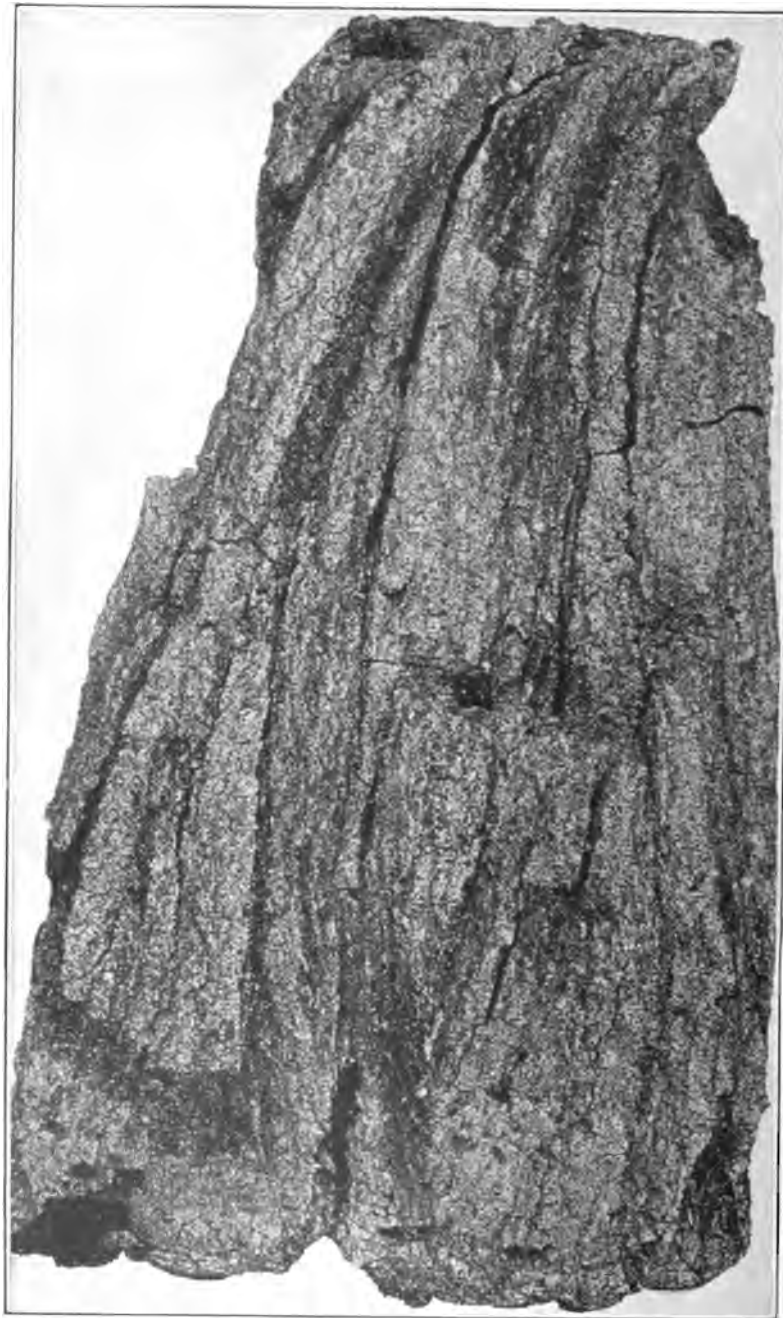
Coke from the mixture was in good-sized pieces and had only a slight tendency to cross fracture. The fusion was good and the coke had a medium cell structure. The porosity was 48.2 per cent and in the shatter test 89.9 per cent was retained on the screen.

Yields per ton of coal, as determined in laboratory, test 14.

	Illinois coal alone.	20 per cent Illinois coal and 80 per cent low-volatile coal.
Coke, per cent.....	69.9	81.3
Tar, gallons.....	6.9	4.1
(NH ₄) ₂ SO ₄ per ton, pounds.....	32.9	20.7
Light oil, pounds.....	26.9	14.8
Gas (including CO ₂ and H ₂ S), cubic feet at 15° C. and 30 inches Hg.....	10, 561	9, 941



COKE FROM 20 PER CENT ILLINOIS (DISTRICT 7, BED 6) COAL AND 80 PER CENT LOW-VOLATILE COAL, TEST 12. IRREGULAR CELL STRUCTURE.



Analysis of coal and coke produced in ovens, test 14.

Constituent.	Illinois coal alone.		20 per cent Illinois coal and 80 per cent low-volatile coal.	
	Coal.	Coke.	Coal.	Coke.
Moisture, per cent.....	7.00		3.10	
Volatile matter, per cent.....	37.85	0.92	21.93	1.96
Fixed carbon, per cent.....	50.80	83.00	68.86	86.41
Ash, per cent.....	11.34	16.08	9.18	11.61
Sulphur, per cent.....	2.82	2.36	1.11	1.01
Phosphorus, per cent.....	.005	.007	.005	.009

TESTS 15 TO 46.

The results of coking tests 15 to 46 are presented in Table 6, following. They are arranged according to districts. Within the district the tests with Illinois coal alone are given first, followed by those with mixtures of Illinois coal and low-volatile coal.

TABLE 6.—Results of

Test No.	Coal tested.	Source of Illinois coal.	Condition of coal.	Quantity of coal used in test.	Number of ovens used in test.	Coking time.
1	2	3	4	5	6	7
15	Illinois.....	Bureau County, district 1, seam 2...	Finely crushed....	<i>Cars.</i> a 80	5	13
16	do.....	Woodford County, district 1, seam 2.....	do.....		5	
17	do.....	District 1, seam 7.....	do.....			6
18	do.....	Tazewell County, district 4, seam 5.....	do.....	a 155	10	12
19	do.....	District 4, seam 5.....	do.....			6
20	do.....	Saline County, district 5, seam 5.....	{Washed screenings finely crushed.}			
21	{75 per cent Illinois; 25 per cent low- volatile.	do.....	do.....			7
22	{60 per cent Illinois; 40 per cent low- volatile.	do.....	do.....			7
23	{60 per cent Illinois; 40 per cent Poca- hontas.	do.....	{Run of mine, fine- ly crushed.}			7
24	Illinois.....	Franklin County, district 6, seam 6.....	Finely crushed.....			
25	do.....	do.....	do.....	1		
26	do.....	do.....	do.....	3		12
27	do.....	do.....	do.....	a 200	13	12-13
28	do.....	do.....	do.....	a 165	11	11½
29	do.....	do.....	do.....	a 80	6	12½
30	do.....	Williamson County, district 6, seam 6.....	do.....	1		7
31	do.....	do.....	do.....	1		6
32	do.....	do.....	do.....	13	12	13
33	do.....	do.....	do.....	a 63		18
34	do.....	do.....	do.....	a 192	13	15
35	do.....	District 6, seam 6.....				6
36	do.....	do.....	Finely crushed.....			7
37	do.....	do.....	do.....			7
38	do.....	do.....	do.....			6
39	do.....	do.....	do.....			6
40	do.....	do.....	do.....			6
41	do.....	do.....	do.....			7

a Tons.

b By-products from oven.

c By-product results represent laboratory test; volume of gas given includes CO₂ and H₂S.

product coking tests 15 to 46.

Breeze.	Gas per ton of 2,000 pounds.	(NH ₄) ₂ SO ₄ per ton.	Tar per ton.	Light oil per ton.	Analysis of coal and coke from ovens.							Physical properties and appearance of coke.	
					Sample.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.	Shatter test: Per cent held on 2-inch screen.	Remarks.
9	10	11	12	13	14	15	16	17	18	19	20	21	22
P.ct	Cu. ft.	Lbs.	Gal	Lbs		P.ct	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	
.....					Coal.....	5.23	40.77	43.62	10.38	4.15		72.3	Coke medium hard and dense.
.....					Coke.....	1.20	21.85	85.60	12.99	1.89		72.3	Coke hard and dense.
.....					Coal.....	8.86	39.84	45.37	5.93	2.07		76.4	Coke hard and dense.
.....					Coke.....	.78	1.03	88.47	9.72	1.06		76.4	Coke hard and dense.
.....					Coal.....	39.76	54.39	5.85	2.14	0.003			Coke hard and dense.
.....					Coke.....								Coke hard and dense.
.....					Coal.....	5.69	38.41	46.03	9.87	2.86		70.0	Do.
.....					Coke.....	.78	1.05	85.62	12.55	1.18		70.0	Do.
.....					Coal.....	40.03	46.98	12.99	4.32	.002			Do.
.....					Coke.....								Do.
.....					Coal.....	3.98	38.02	50.60	7.40	1.88			Coke light in weight, smoky,
.....					Coke.....	.38	1.98	84.74	12.90	1.40			and black; entirely fused.
.....					Coal.....	3.00	33.84	55.66	7.50	1.78	.010		Coke porous, with few cross
.....					Coke.....	.28	1.52	86.20	12.00	1.34	.014	66.0	fractures.
.....					Coal.....	3.71	32.75	55.94	7.60	1.73	.016		Coke porous; of poor appear-
.....					Coke.....	.25	1.49	86.96	11.30	1.16	.017	68.0	ance.
.....					Coal.....	2.01	28.77	58.31	10.91	1.57			Coke fairly tough; good look-
.....					Coke.....	.35	2.51	81.76	15.38	1.26		75.1	ing; not foundry grade.
.....					Coal.....								Coke porous and fingerlike.
.....					Coke.....								Coke light, porous, and fragile.
.....					Coal.....	5.07	34.52	49.76	10.65	1.51	.007		Coke light, porous, and fragile.
.....					Coke.....	.85	2.51	80.24	16.40	.93	.015		Coke light, porous, and fragile.
.....					Coal.....	7.53	32.17	50.46	9.84	1.94			Coke hard and dense.
.....					Coke.....	.10	.21	86.12	13.57	1.23		73.0	Coke hard and dense.
.....					Coal.....	7.48	34.52	50.84	7.16	1.14			Do.
.....					Coke.....	.48	.92	88.47	10.13	.81		76.0	Do.
.....					Coal.....	6.76	35.04	50.43	7.77	1.19		75.9	Do.
.....					Coke.....	.36	.64	88.92	10.08	.78		75.9	Do.
.....					Coal.....	5.08	35.78	49.61	9.73	2.66		68.4	Do.
.....					Coke.....	1.18	.32	86.16	12.34	1.38		68.4	Do.
.....					Coal.....	2.08	27.13	60.67	10.12	1.28			Coke coarse and spongy.
.....					Coke.....								Coke coarse and spongy.
.....					Coal.....	4.16	34.18	50.42	11.24	1.84			Coke hard, coarse, and inferior.
.....					Coke.....	.19	1.23	82.61	15.97	1.39			Coke hard, coarse, and inferior.
.....					Coal.....	5.62	35.58	48.89	9.91	1.53			Coke hard and dense.
.....					Coke.....								Coke hard and dense.
.....					Coal.....	6.23	41.77	42.62	9.38	3.97		76.0	Do.
.....					Coke.....	.28	1.07	84.27	14.38	2.16			Do.
.....					Coal.....	5.17	32.33	53.14	9.36	1.43		73.2	Do.
.....					Coke.....								Do.
.....					Coal.....	3.00	38.15	48.49	13.36	4.09	.025		Do.
.....					Coke.....								Do.
.....					Coal.....	5.67	32.89	53.77	13.34	1.45	.049		Coke small in size, light in
.....					Coke.....								weight; had open cell struc-
.....					Coal.....								ture.
.....					Coke.....								Coke small in size, light in
.....					Coal.....	35.39	56.00	8.61	1.17	.007			weight; had open cell struc-
.....					Coke.....								ture.
.....					Coal.....	3.33	35.00	56.80	8.20	2.02	.025		Coke soft; fingerlike; open cell
.....					Coke.....								structure.
.....					Coal.....	2.00	39.07	49.23	11.70	1.17	.004		Coke had poor irregular cell
.....					Coke.....								structure; considerable non-
.....					Coal.....	2.76	81.46	15.78	1.05	.005	76.3		fused material.
.....					Coke.....								Coke fingerlike and brittle;
.....					Coal.....	3.00	31.84	52.32	15.84	3.25	.027		showed spongy material.
.....					Coke.....								Coke fair size; rough surfaces;
.....					Coal.....	2.33	35.58	52.86	11.56	2.18	.023		considerable nonfused ma-
.....					Coke.....								terial.

* By-products determined by laboratory test.

* Results represent averages from two trials.

TABLE 6.—*Results of by-pro*

Test No.	Coal tested.	Source of Illinois coal.	Condition of coal.	Quantity of coal used in test.	Number of ovens used in test.	Caking time.
1	2	3	4	5	6	7
42	{ 85 per cent Illinois; 15 per cent Durham.	Franklin County, district 6, seam 6..	(a)	<i>Care.</i>	<i>Hrs.</i>	
43	{ 20 per cent Illinois; 80 per cent low-volatile.					
44	do.....	District 6, seam 6.....	Finely crushed.....			
45	Illinois.....	District 8, seam 6.....	do.....			
46	{ 20 per cent Illinois; 80 per cent low-volatile.	District 8, seam 6.....	do.....			

(a) 74.6 per cent passed through $\frac{1}{8}$ -inch screen.

coking tests 15 to 46—Continued.

Freeze.	Gas per ton of 2,000 pounds.	(NH ₄) ₂ SO ₄ per ton.	Tar per ton.	Light oil per ton.	Analysis of coal and coke from ovens.							Physical properties and appearance of coke.	
					Sample.	Moisture.	Volatile matter.	Fixed carbon.	Ash.	Sulphur.	Phosphorus.	Shatter test: Per cent held on 2-inch screen.	Remarks.
9	10	11	12	13	14	15	16	17	18	19	20	21	22
P.ct.	Cu. ft.	Lbs.	Gal.	Lbs.		P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	P.ct.	
.....					{Coal.....	8.80	34.67	57.42	7.91	1.33			{Coke hard and dense; slight tendency to form fingers.
.....					{Coke.....	5.04	2.52	84.69	12.79	1.09			
.....					{Coke.....	.76	20.78	69.01	9.45	1.06			
.....					{Coal.....	4.80	21.09	70.79	8.12	.80	.005		{Vertically cross fractured; cell structure rather irregular; fusion complete.
.....					{Coke.....	1.29	87.02	11.15	.76	.007	.76.2		
.....					{Coke.....	2.20	41.47	42.71	11.32	2.76	.009		
.....					{Coal.....	5.22	79.21	15.57	2.57	.013	.77.2		{Coke easily reduced to powder; had no cell structure.
.....					{Coke.....	21.86	69.62	8.52	1.03	.007			
.....					{Coke.....	3.00	85.75	11.25	.61	.006	.82.8		
.....					{Coal.....	21.86	69.62	8.52	1.03	.007			{Coke irregular and cross fractured; poor cell structure; nonfused material.
.....					{Coke.....	3.00	85.75	11.25	.61	.006	.82.8		
.....					{Coke.....	3.00	85.75	11.25	.61	.006	.82.8		

^b By-products determined by laboratory test; volume of gas given includes CO₂ and H₂S; composition of gas was as follows, values representing percentages: CO₂, 1.8; illuminants, 1.9; O₂, 0.4; CO, 4.7; CH₄, 25.0; H₂, 61.5; N₂, 4.9.

^c By-product results represent laboratory test; volume of gas given includes CO₂H₂S.

TESTS OF ILLINOIS COALS IN GAS RETORTS.

Although Illinois coals have been used for making gas for several years, their use has been confined to some of the smaller plants and has not grown rapidly. In order to determine the suitability of Illinois coals for gas manufacture, tests have been made from time to time, and the results of some of the tests are given in the following pages.

At present, plants using Illinois coal only are situated at Centerville, Mount Vernon, and Duquoin, Ill., and at St. Charles, Booneville, Cape Girardeau, and Clinton, Mo. Plants using part Illinois coal and part Eastern gas coal are given on page 63.

TESTS AT ST. LOUIS, MO.

Tests on a large scale under actual operating conditions made by a gas company at St. Louis showed the yields that can be expected from Illinois coals in commercial practice. The duration of the tests was sufficient to show any difficulties in operation due to the nature of the coal and to give average results. The products were systematically sampled and carefully analyzed during the tests, and all figures given are average results obtained from several samples.

TEST 1.

Test 1 was a six-day test of coal from Perry County, district 6, seam 6. The coal was of lump and egg size. The lump contained more slate and iron pyrites than the egg, which appeared to be comparatively clean. The coal as mined contained a white substance, probably calcium carbonate or calcium sulphate, as a thin parting.

The coal was carbonized in a retort house of 32 benches of fire, direct fired. During the first three days about one half of the number of active retorts were charged with 500 pounds each and the other half were charged with 525 pounds each. The last three days the charges were 525 and 550 pounds to the retort. The period of carbonization was $4\frac{1}{2}$ hours. Several charges of 500 pounds each were carbonized for 6, 8, and 12 hours, and 1 charge of 1,000 pounds was carbonized for 12 hours in order to show the effect on the coke. No trouble was experienced in the retort house while the coal was being carbonized.

The average yield of gas, corrected to 60° F. and 29.5 inches of mercury, was 3.92 cubic feet per pound of coal. The crude gas contained 1.2 grains of hydrogen sulphide per 100 cubic feet at the outlet of the exhaustor and 369 grains at the outlet of the scrubbers. The average organic sulphur was 11.6 grains. The quality of the gas was good, but the yield was low. The daily (24-hour) production of gas, and its quality, were as follows:





COKE FROM 100 PER CENT ILLINOIS (DISTRICT 6, BED 6) COAL, TEST 14. GOOD FUSION, SLIGHTLY IRREGULAR CELL STRUCTURE.

Data regarding daily production of gas in test 1.

Item.	First day.	Second day.	Third day.	Fourth day.	Fifth day.	Sixth day.
Quantity of gas made, corrected to 60° F. and 29.5 inches Hg, thousand cubic feet.....	1,202	1,242	1,195	1,207	1,229	1,164
Coal carbonized, pounds.....	302,500	306,000	307,300	313,500	314,400	302,200
Average yield of gas per pound of coal, cubic feet..	3.97	4.05	3.89	3.85	3.91	3.85
Candle power of 24-hour sample.....	14.7	14.6	13.5	12.0	14.7	12.9
Gross heating value of 24-hour sample, B. t. u.....	617	604	600	592	617	601

Several carefully weighed charges were carbonized for 4½ hours and the coke weighed hot to determine the yield. An average of 56 per cent was obtained. The coke was light and porous and contained considerable unfused material. It was of poor grade. The percentage of breeze was high. The coke was tried for the production of water gas in two 11-foot generators. Its action was not as satisfactory as that of Pittsburgh coke, the yield for the six days being about 10 per cent less than the usual yield with the Pittsburgh coke. The difference was due to the greater amount of clinkers and the increased time required for removing them. The daily record of the water gas test follows:

Data regarding production of water gas from coke produced in test 1.

Item.	First day.	Second day.	Third day.	Fourth day.	Fifth day.	Sixth day.
Quantity of gas made, corrected to 60° F. and 29.5 Hg, thousand cubic feet.....	4,351	4,296	4,045	4,260	5,598	5,706
Time on clinkers and breakups, hours and minutes.....	4 49	5 15	6 8	5 10	8 6	7 56
Coke per thousand cubic feet of gas made, pounds.....	36.5	35.8	37.1	38.4	37.3	37.6

No determination was made of the quantity of tar produced, but samples taken at the overflow of the hydraulic main were examined for appearance and specific gravity of the tar and for the free carbon contained in it. It was thin and had a tendency to run through connections; the specific gravity was 1.17 and the free carbon was 10 per cent. The average yield of ammonia (NH₃) was 5.21 pounds per ton of coal. The strength of liquor in the various storage wells was not determined.

Summary of results of test 1.

Weight of coal carbonized, tons.....	922.9
Duration of test, days.....	6
Gas per pound of coal, cubic feet.....	3.92
Coke, per cent.....	58
Ammonia per ton of coal, pounds.....	5.21
Coke per 1,000 feet of water gas, pounds.....	37.1
Candlepower of gas.....	13.7
Heating value of gas, B. t. u.....	605

<i>Average composition of gas by volume, test 1.</i>		
Constituent.		Per cent.
CO ₂		2.9
Illuminants.....		4.2
O ₂		1.4
CO.....		8.9
CH ₄		29.4
H ₂		42.1
N ₂		11.1

Average analysis of coal and coke, test 1.

	Coal.	Coke.
Moisture, per cent.	6.53	
Volatile matter, per cent.....	36.42	5.64
Fixed carbon, per cent.....	47.48	79.07
Ash, per cent.....	9.57	15.29
Sulphur, per cent.....	1.00	.76
Heating value, B. t. u.....	12,148	12,367

TEST 2.

In test 2 "egg" coal from Franklin County, district 6, seam 6, used.

The coal was carbonized in four benches of four of the direct-fired type. At the start 525-pound charges and 4½-hour periods were used. These were first changed to 600 pounds and 6 hours, then to 700 pounds and 7 hours, and later to 800 pounds and 8 hours. In the first three combinations the coal was well carbonized; with the last combination a considerable amount of gas remained in the retorts at the end of the period. Subsequently charges of 700 pounds and an 8-hour period were used. Because the supply of coal was small for a test with all of the retorts, the yield of gas was determined by a laboratory method. For this purpose a carefully selected sample of coal was taken from each car as it was unloaded. Analyses of these samples were made and the composition and the yield of gas were determined in a laboratory retort. Five cubic-foot samples of gas were also taken from the hydraulic main, near the center of each of four benches, in order to determine the composition of the foul gas from the retorts. The composition of this gas and that obtained from the laboratory retort are given in the summary of results of test.

The coke obtained with the first four combinations of charge and coking period was soft and porous, there being no noticeable difference caused by changes in weight of charge and time of carbonization.

The coke from the 700-pound charge and 8-hour period seemed slightly harder and denser than that from the other combinations. The coke was tested in a water-gas generator, and made a large amount of clinker, owing to fusion of the excessive amount of ash. The amount of clinkering was increased from 25 to 30 minutes per watch.

steam pressure was increased 2 pounds above that ordinarily used with Pittsburgh coke, in order to soften the clinker, but the increase of steam pressure lowered the heat in the generator and materially decreased the gas production. More Illinois coke was required than Pittsburgh during an equal period of time.

Summary of results of test 2.

	Results from work- ing bench.	Results from labora- tory retort.
Coke, including breeze, per cent.....	57.1	61.3
Gas per pound of coal, cubic feet.....		5.32
Candlepower of gas.....		17.3
Heating value per cubic foot of gas, B. t. u.....	661	632
H ₂ S per 100 cubic feet of gas, grains.....	1,235	
Organic sulphur per 100 cubic feet of gas, grains....	26	

Average composition of gas, test 2.

	Results from work- ing bench.	Results from labora- tory retort.
CO, per cent.....	2.2	3.4
Illuminants, per cent.....	3.8	5.4
O ₂ , per cent.....	.5	.7
CO ₂ , per cent.....	8.5	10.4
CH ₄ , per cent.....	34.8	30.3
H ₂ , per cent.....	46.2	43.3
N ₂ , per cent.....	3.0	6.3

Average analysis of coal and coke, test 2.

	Coal.	Coke.
Moisture, per cent.....	3.55	1.12
Volatile matter, per cent.....	37.55	4.82
Fixed carbon, per cent.....	48.63	76.55
Ash, per cent.....	10.42	17.51
Sulphur, per cent.....	2.32	2.46
Heating value, B. t. u.....	12,392	11,502

TEST 3.

Test 3 was a 15-day test of coal from Perry County, district 6, seam 6. The coal contained considerable "slate," much of it being in thin flat chunks.

The test was made with four benches of fours of the direct-fired type. The charges were 500 pounds each, and the time of carbonization was 4½ hours during the first part of the test and 6 hours during the later part. Two retorts were charged with 1,000 pounds each, a 14-hour period of carbonization being used. Gas analyses were made daily. The tar and ammonia wells were measured before and after the test and before and after each time either tar or ammonia was pumped away. Samples of ammonia liquor were taken at the same time.

The yield of gas, corrected to 60° F. and 29.5 inches of Hg. 4.136 cubic feet per pound of coal. The low yield can be accounted for to some extent by the high ash content of the coal. The quality of the gas was good, comparing well with that from Pittsburgh.

The coke from two retorts of the same bench was weighed. The average yield, including breeze, was 60.2 per cent, which is 1 per cent less than the yield from Pittsburgh coal. The coke was light and friable and contained a number of unfused places which seemed to prevent the formation of large pieces. It was tested as fuel under the benches, where it furnished satisfactory heat and did not form a hard clinker. The ash content was high.

The ammonia was 6.42 pounds per ton of coal carbonized, somewhat higher than from Pittsburgh coal. The tar averaged 10.65 gallons per ton of coal, which is considerably less than is obtained from Pittsburgh coal.

Summary of results of test 3.

Weight of coal carbonized, tons.....	226.7
Duration of test, days.....	12
Gas per pound of coal, cubic feet.....	4.136
Coke, per cent.....	54.4
Breeze, per cent.....	5.8
Ammonia per ton of coal, pounds.....	6.42
Tar per ton of coal, gallons.....	10.65
Coke fired per ton of coal carbonized, pounds.....	251
Candlepower of gas.....	15.6
Heating value per cubic foot, B. t. u.....	656
H ₂ S per 100 cubic feet, grains.....	585

Average composition of gas, test 3.

Constituent.	Per cent.
CO ₂	1.9
Illuminants.....	4.4
O ₂	.5
CO.....	8.0
CH ₄	40.0
O ₂	40.1
N ₂	5.1

Average analysis of coal and coke, test 3.

Constituent.	Coal.	4½-hour coke.	6-hour coke.	1-hour coke.
Moisture, per cent.....	4.36			
Volatile matter, per cent.....	34.18	6.17	4.97	
Fixed carbon, per cent.....	48.69	80.23	79.24	
Ash, per cent.....	12.77	13.60	15.79	
Sulphur, per cent.....	1.17	.92	.77	
Heating value, B. t. u.....	11,808	11,939	11,451	

TEST 4.

Test 4 was a 24-hour run with 6-inch coal from Franklin County, district 6, seam 6.

The coal was carbonized in a retort house of 32 benches of fours, direct fired. Charges of 450 and 500 pounds to the retort were made during the day and charges of 400 and 450 pounds during the night because the supply of coal was not sufficient to make the larger charges for 24 hours. The time of carbonization was 4½ hours. Three charges of 500 pounds each were carbonized for 6, 8, and 12 hours, and one charge of 1,000 pounds for 12 hours for the purpose of observing the resultant coke.

The average yield of gas, corrected to 60° F. and 29.5 inches of Hg., was 4.06 cubic feet per pound of coal. The crude gas contained 604 grains of hydrogen sulphide per 100 cubic feet at the outlet of the exhauster and 505 grains at the outlet of the scrubbers. The quality was good, but the yield was low compared with the values for Eastern gas coals.

The yield of coke, including breeze, was 60 per cent. The quality was poor, being similar to that from the other tests. With the first three combinations of charge and coking time no difference could be observed on account of the longer periods of carbonization. The coke from the 12-hour charge appeared to be somewhat improved, but unfused pieces were noticeable. The coke was tried in water-gas generators, with results similar to those obtained with the coke from other samples of Illinois coal. A longer time was required to remove clinkers and a larger quantity of fuel was used per 1,000 feet of gas than with Pittsburgh coke.

Summary of results of test 4.

Weight of coal carbonized, tons.....	140.9
Duration of test, hours.....	24
Gas per pound of coal, cubic feet.....	4.06
Coke, including breeze, per cent.....	60
Ammonia at outlet of exhauster, pounds.....	4.18
Coke per 1,000 feet of water gas, pounds.....	36.5
Candlepower of gas.....	16.7
Heating value of gas, B. t. u.....	644

Average composition of gas, test 4.

Constituent.	Per cent.
CO ₂	2.4
Illuminants.....	3.6
O ₂	0.5
CO.....	9.0
CH ₄	32.9
H ₂	44.1
N ₂	7.5

Average analysis of coal and coke, test 4.

	Coal.	Coke.
Moisture, per cent.....	4.57.....	
Volatile matter, per cent.....	36.35	4.78
Fixed carbon, per cent.....	49.25	83.44
Ash, per cent.....	9.83	11.78
Sulphur, per cent.....	1.37	0.73
Heating value, B. t. u.....	12,621	12,698

TEST AT ANN ARBOR, MICH.

In illuminating-gas investigations commenced by the technical branch of the United States Geological Survey in 1908 effort was made to ascertain the possibility of using for gas making other cheaper fuels than Eastern coals. The work was carried on at Ann Arbor, Mich., at the experimental plant of the Michigan Gas Association. Alfred H. White, professor of chemical engineering at the University of Michigan, directed the work and was in charge of the investigations. The fuel-testing work was subsequently transferred to the Bureau of Mines and the report of the tests published by that bureau.^a

Among the coals tested was a sample of $\frac{1}{4}$ -inch size from No. 4 mine at Harrisburg, Saline County, Ill., district 5, seam 1.

The test was made in a single retort of the standard $\square$ size which was one of the upper retorts of a bench of six with two-quarter depth regenerative setting. This retort was equipped with a complete system for cleaning the gas and measuring and sampling the products independently of the other five. A charge of 100 pounds was carbonized for about $4\frac{1}{2}$ hours. The coke produced was dark and easily broken. The yield of gas was 4.3 cubic feet per pound of coal, with an average candlepower of 15.2. The heating value of the gas was 632 B. t. u. per cubic foot and 2,718 B. t. u. per pound of coal. The ammonia yield was large, being 6.38 per cent per ton of coal, or 19 per cent of the theoretical. The amount of naphthalene was extremely small, probably because of the low retort temperature. The hydrogen sulphide in the gas at the outlet of the scrubber was 50 per cent more than in gas from Pittsburgh coal.

The results in the summary following are on the basis of coal charged to the retort.

^a See White, A. H., and Barker, Perry, Coals available for the manufacture of illuminating gas: Bureau of Mines, 1911, 77 pp.

Summary of results of coking test of Harrisburg coal at Ann Arbor, Mich.

Weight of coal carbonized, pounds.....	400
Gas, per cent.....	14.3
Coke, per cent.....	62.3
Tar, per cent.....	8.6
Ammonia liquor, per cent.....	6.0
Unaccounted for, per cent.....	8.8
Gas per pound of coal, cubic feet.....	4.3
Handpower.....	15.2
Gross heating value per cubic foot, B. t. u.....	632
Ammonia per ton of coal, pounds.....	6.38
Tar per ton of coal, pounds.....	166
Naphthalene per ton of coal, pounds.....	4.5
Hydrogen sulphide at outlet of scrubbers per 100 cubic feet of gas, grains.....	1,390
Maximum temperature in retort, ° F.....	1,360
Average temperature, exterior of retort, ° F.....	1,779

Average composition of gas from Ann Arbor test.

Constituent.	Per cent.
O ₂	2.5
Illuminants.....	3.7
H ₂	.8
CO.....	7.9
H ₄	33.5
CH ₄	45.6
H ₂	6.0

Average analysis of coal and coke, Ann Arbor test.

	Coal as charged.	Dry coke.
Moisture, per cent.....	4.66	
Volatile matter, per cent.....	34.44	2.22
Fixed carbon, per cent.....	53.71	85.48
Ash, per cent.....	7.19	12.30
Sulphur, per cent.....	1.96	2.08
Heating value, B. t. u.....	12,919	12,427

TEST AT DE KALB, ILL.

Test of coal from Perry County, district 6, seam 6, made at De Kalb, Ill., gave the following results:

Weight of coal, pounds.....	20,600
Gas, obtained, cubic feet.....	93,400
Yield of gas per pound of coal, cubic feet.....	4.53
Coke, including breeze, pounds.....	13,800
Coke, per cent.....	56
Breeze, per cent.....	11

Composition of gas.

CO ₂ , per cent.....	2.4
Illuminants, per cent.....	3.5
O ₂ , per cent.....	1.1
CO, per cent.....	10.6
CH ₄ , per cent.....	30.8
H ₂ , per cent.....	43.9
N ₂ , per cent.....	8.2
Heating value, B. t. u.....	554

Analysis of coal and coke.

	Coal.	Coke.
Moisture, per cent.....	9.0	0.5
Volatile matter, per cent.....	36.0	1.0
Fixed carbon, per cent.....	46.0	85.5
Ash, per cent.....	9.0	13.0
Sulphur, per cent.....	.85	1.04

USE OF ILLINOIS COAL WITH EASTERN GAS COAL.

During the past two years the practice of using one-third Illinois coal and two-thirds Eastern gas coal has been adopted at some plants. The customary method of operating with two coals is to charge one-third of the retorts in a bench with Illinois coal and other two-thirds with Eastern gas coal. In this way the coke from Illinois coal can be kept separate from that from the Eastern gas coal and can be used for heating the retorts and for the production of water gas, and the coke from Eastern gas coal can be sold for domestic use. In a few cases the coals have been mixed before being charged into the retorts.

The saving resulting from this practice varies with the comparative cost of the coals which depends largely on freight rates. When Eastern gas coal costing \$1 a ton more than Illinois coal, the use of one-third Illinois coal represents a saving of about 2 cents per 1,000 feet of gas. Following are given comparative data for a plant making about 200,000 feet of gas daily, showing results when Eastern gas coal is used alone and when it is used in combination with one-third Illinois coal.

The figures give the cost of fuel and labor per 1,000 cubic feet of gas and do not include fixed charges.

General cost data for fuel and labor in a gas plant making about 200,000 cubic feet of gas daily.

COST WHEN EASTERN GAS COAL WAS USED.

1,374,600 pounds of coal at \$3.20 per ton, used in 31 days.....	\$2, 199. 36
220,336 pounds of coke at \$4 per ton.....	440. 67
Labor.....	650. 71
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WHEN ONE-THIRD ILLINOIS COAL AND TWO-THIRDS EASTERN GAS COAL
WERE USED.

2,352,779 pounds of Eastern coal at \$3.20 per ton, used in 80 days.....	\$3,764.45
322,620 pounds of Illinois coal at \$2.22 per ton.....	913.11
528,741 pounds of coke at \$4 per ton.....	1,057.48
Labor.....	1,310.76
	7,045.80

Detailed cost data.

Coal used.	Yield of gas, 1,000 cubic feet.	Yield per pound.	Heat- ing value.	Average cost.				Average credit coke and tar.	Average net cost per 1,000 cubic feet.
				Coal.	Bench fuel.	Labor.	Total.		
and Illinois.....	6,737 15,648	Cu. ft. 4.90 4.93	B. t. u. 587 591	\$0.325 .290	\$0.065 .067	\$0.096 .084	\$0.487 .450	\$0.269 .268	\$0.218 .182

RESULTS OF PLANT TESTS.

following data concerning the use of coal from Perry County, at 6, seam 6, when used with Eastern gas coal, were furnished by Mr. Edward H. Taylor, fuel and gas engineer. They are the average results of several days' operation at the plants given. The figures in the last column represent the product of the yield of gas per pound of coal and the heating value of the gas per cubic foot. This product gives the thermal value of the gas obtained from 1 pound of coal, and furnishes a good basis for comparing results:

7.—Results of using in different gas plants one-third Illinois coal and two-thirds Eastern gas coal.

Location of gas plant. ^a	H ₂ S in gas at outlet of ex-hauster.	H ₂ S in gas at outlet of scrubbers.	Total sulphur in purified gas.	Yield of gas per pound of coal.	Heating value of gas per cubic foot, B. t. u.	Heating value of gas per pound of coal, B. t. u.
	Grains per 100 cubic feet.	Grains per 100 cubic feet.	Grains per 100 cubic feet.	Cubic feet.		
.....	330	240	13.2	4.5	615	2,767
.....	390	275	11.9	4.6	608	2,997
.....			16.3	5.22	578	3,016
.....	280	210	9.9	5.0	591	2,955
.....	310	270		4.9	575	2,818
.....	280	270	15.0	5.01	625	3,131
.....	270	200	17.2	4.9	606	2,969
.....	180	150	11.2	4.6	598	2,751
.....				5.0		
.....	230	180	10.3	4.6	609	2,801
.....	260	190		5.01	590	2,956
.....				4.8		
.....	300	210	10.2	4.73	603	2,852
.....				4.41	596	2,628
.....	400	220	14.8	5.06	595	3,011
.....	240	190	10.0	5.17	590	2,999
.....	303		8.6	5.24	598	3,134
.....	402	388	11.2	5.3	595	3,153

^aAll situated in Illinois.

^bAt this plant one-fourth instead of one-third Illinois coal was used.

WATER GAS FROM ILLINOIS COAL.

A gas company in East St. Louis is using Illinois coal for manufacturing water gas. Run-of-mine coal, instead of coke, is charged to generators. The company is making about 1,000,000 feet of gas a day by this method, and reports satisfactory results. Approximately 37 pounds of coal is used per 1,000 feet of gas, and the gas obtained is reported to have a heating value about 75 B. t. u. per cubic foot higher than that from coke. The coal used is from Jackson County, district 2, seam 2.

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DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

CONTROL OF HOOKWORM INFECTION
AT THE DEEP GOLD MINES OF THE
MOTHER LODE, CALIFORNIA

BY

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PREFACE.

The Federal Bureau of Mines is authorized by law to make, under direction of the Secretary of the Interior, investigations with a view to improving conditions that impair the health of miners. Prevention of diseases peculiar to special industries is occupying much attention to-day. In the past possibly too little attention was given to certain baneful effects that were growing up side by side with great industries. Successful combating of industrial diseases is a somewhat complex problem, as it requires a knowledge of the industry and of the disease.

The Bureau of Mines in its health investigations seeks to cooperate with the existing Federal or State health agencies. In the investigation of hookworm infection in the deep gold mines of the Mother Lode in California, described in this report, three agencies were represented—the California State Board of Health, the Industrial Accident Commission of the State of California, and the Federal Bureau of Mines.

Hookworm infection is a communicable disease; hence the State Board of Health was desirous of stamping it out. Hookworm infection in mines might be regarded as an industrial injury, so the Industrial Accident Commission of the State of California was anxious of eradicating it. Hookworm infection is prevalent among miners, and as infected miners may spread the infection from the mines of one State to those of another, the Federal Bureau of Mines was interested in preventive measures. The largest share of the expense of the investigation described was borne by the California State Board of Health; most of the field work was done by the Bureau of Mines.

In this investigation Dr. James G. Cumming, director of the State Bureau of communicable diseases, represented the California State Board of Health; Edwin Higgins, chief mine inspector, represented the Industrial Accident Commission of the State of California; and Joseph H. White, sanitary engineer, represented the Bureau of Mines.

VAN. H. MANNING, *Director.*



CONTROL OF HOOKWORM INFECTION AT THE DEEP GOLD MINES OF THE MOTHER LODE, CALIFORNIA.

By JAMES G. CUMMING and JOSEPH H. WHITE.

INTRODUCTION.

This report presents the results of an investigation of hookworm infection in the deep gold mines of the Mother Lode, California. In this investigation 1,440 miners of the Mother Lode have been examined for hookworm infection; 444 miners were found to be infected; and 91 miners have taken the treatment and have gotten rid of the infection. Within a short time after this preliminary examination, an industrial strike started, which gradually scattered the miners and made it difficult to institute the treatment of many of those who had been found infected. With the completion, however, of the State survey all these will eventually be identified and treated.

In 1906 Dr. F. F. Sprague, of Jackson, Cal., and formerly of the Philippine Islands, recognized the prevalence of hookworm infection in the gold mines of California. He treated many cases of the infection and brought the prevalence of the disease to the attention of his colleagues. Following closely this pioneer work of Dr. Sprague was that of Dr. E. E. Endicott, health officer of Amador County. In 1911 Dr. Endicott^a related his experience in a paper before the American Medical Association.

In 1909, Dr. Herbert Gunn, under the auspices of the California State Board of Health, made an investigation to determine the prevalence of the disease among miners.^b He reported on the blood and stool examinations of groups of men from three large mines in Amador County. The high eosinophilia verified by stool examinations in many of the cases, and the demonstration of larvae in the feces from the workings led Dr. Gunn to the following conclusions:

Hookworm disease is endemic in certain mines of California.

^aEndicott, E. E., My experiences with hookworm infection in the deep gold mines of California: Jour. Am. Med. Assoc., vol. 57, Sept. 30, 1911, pp. 1106-07.

^bGunn, H., Hookworm disease in the gold mines of California: Monthly Bull. California State Board of Health, vol. 6, December, 1910, p. 408.

2. From 50 to 80 per cent of those working in these mines are infected.

3. The infection undoubtedly is present in practically all the gold mines of California and in those of Nevada just over the border.

When it was definitely determined by the California State Board of Health to make an investigation, in cooperation with the Industrial Accident Commission of the State of California and the Federal Bureau of Mines, as to the prevalence of hookworm infection among the miners of California, the following procedure was planned: (1) To encourage the superintendents of the various mines to cooperate in the campaign; (2) to diagnose fecal specimens from the miners and determine the percentage of infected miners; (3) to reach an agreement with the superintendents that all infected men must be treated; (4) to make reexaminations of all treated men; (5) to issue "hookworm certificates"; and (6) to eventually reexamine all California miners.

A visit to several of the mines on the Mother Lode made it evident that the mine operators would cooperate most cordially in this campaign; in fact, several superintendents had already considered the advisability of such an undertaking.

ACKNOWLEDGMENTS.

The authors of this paper owe much to the helpful cooperation of the mining operators of the Mother Lode and to all their coworkers in the California State Board of Health; the Industrial Accident Commission of the State of California. and the Federal Bureau of Mines.

DESCRIPTION OF THE MOTHER LODE MINING DISTRICT.

The Mother Lode consists of a series of veins of gold-bearing quartz, situated in Eldorado, Amador, Calaveras, Tuolumne, and Mariposa counties, California. These veins lie in a belt about 6½ miles wide and about 70 miles long that has a general northwesterly direction, and the veins dip about 70° or 80° northeast. The Kennedy Mining & Milling Co. mines the ore through a vertical shaft 4,000 feet deep, and has the deepest gold mine in the United States. Just outside this belt gold was discovered in 1848. At first placer mining in the creeks engaged everyone's attention, but some enterprising miners sought the source of the golden sands, and quartz mining started in the early fifties. The series of rich gold-bearing quartz veins was appropriately named the Mother Lode. This district has yielded many millions of dollars' worth of gold since the pioneering days of '49, and as it yielded \$5,788,504 during 1915, an increase of \$712,952 over 1914 the district evidently still has a long life before it.

SERIOUSNESS OF HOOKWORM INFECTION IN THE DISTRICT.

any material prosperity in store for this mining district can not be on a stable basis if about 40 per cent of the gold miners working underground contract an insidious infection. The authors have no intention to raise imaginary evils. It is frankly admitted that some of the men found infected looked healthy, and if the diagnosis were made by symptomatology, instead of by actual microscopic examination of stools, the presence of the infection would not have been surmised. The infection, although widespread, is at present, in many respects, not far advanced. Unfortunately the obtaining of a personal history of the large number of men found infected was impracticable. Had such histories have been procured it would have been found that the infection produced various effects. The hookworm is an insidious parasite. It does not actually kill, but extends its evil influence in a quiet, cumulative manner, and in different directions. A man can not enjoy normal health while the worms are attached to the inside of his intestines and daily sucking his blood. This fact is illustrated by the testimony as to improved physical condition given by some of the miners after taking the hookworm treatment. It should be remembered that although a miner with hookworm may not be seriously affected himself there is always danger of his spreading the infection if he commits a nuisance in a place favorable to the development of the eggs.

ECONOMIC LOSS DUE TO HOOKWORM.

An attempt was made to determine the economic loss due to hookworm infection in this district. Conclusive evidence was difficult to obtain on account of the many factors involved, chief among these being the lack of permanent mine crews, the miners coming and going, so that records extending over a long period were not available. The time books of the mine enjoying the most permanent labor force were carefully studied for the year 1915. At this mine 80 of the miners and muckers had been in the employ of the company for the entire year. Of these 80 men, 42 were infected and 38 were free from infection. Discounting Sunday absences and absences on account of accidents, it was found that the men infected with hookworm were absent, on an average, nine days more during the year 1915 than the uninfected men. Also it was found that the number of seven-day absences among those with hookworm disease was 57 per cent greater than among those without hookworm. At \$3 a day, the annual loss to the miner for nine days' idleness is \$27; the loss to the operator can only be approximated. Another point to be considered is this: If the miner stayed away from work for nine days,

how many days did he report for duty in a condition unfit to do a good day's work? In a few special cases men that were sick and anaemic were found to be infected, and after treatment immediately showed signs of improvement. Although not directly measurable, the loss caused by hookworm to the miners and mine operators of the Mother Lode has without doubt amounted to many thousands of dollars.

GENERAL CONDITIONS IN THE MINES.

The conditions of the Mother Lode mines and how they compare with conditions in other mining districts of the United States are discussed in some detail on pages 9 to 35, but in general it may be said that the sanitary conditions are believed to be above the average. A number of improvements indicative of a progressive and liberal-minded policy can be found in the mines of the Mother Lode. Among these might be mentioned the presence of miners' wash change houses, underground toilet accommodations, garbage receptacles underground for food remnants, water for washing the hands before eating and benches for the miners to sit on while eating, bubbling drinking-water fountains underground, filtered drinking water cooled during the summer, water drills and sprinkling devices in the mines, regulations against firing shots between shifts, respirators supplied to miners free of charge, dust-settling devices on the grinders in the mills, improved drainage conditions in the mines, Sunday a holiday, and vacations with pay granted to the men. It must be understood that the equipment and the privileges named are not found at all of the mines, the list representing the sum total rather than the universal condition. One of the improvements may be found at one mine, another at a second mine, and so on. The presence of these improvements within the district makes their lack in several of the mines more evident. If one mine can install these improvements their practicability has been proved; why not all the mines?

HOOKWORM INFECTION IN THE MINES.

By JOSEPH H. WHITE.

MICROSCOPIC EXAMINATIONS.

Fecal specimens from 1,096 of the workingmen from eight Mother mines were microscopically examined for hookworm eggs; results are shown in Table 1. These data, with other information collected about the 1,096 workingmen, furnish the basis of this report.

TABLE 1.—Results of examining fecal specimens of 1,096 workmen employed at eight mines in the Mother Lode district.

Mine.....	A.	B.	C.	D.	E.	F.	G.	H.	Total.
Number of men examined.....	195	292	199	70	56	47	138	99	1,096
Number found infected.....	66	47	107	25	13	8	44	27	337
Percentage infected.....	34	16	54	36	23	17	32	27	31
Underground workers:									
Number examined.....	158	292	171	57	56	29	121	85	969
Number found infected.....	95	47	114	22	13	6	40	26	363
Percentage infected.....	60	16	67	39	23	21	33	31	38
Surface workers:									
Number examined.....	37		28	13		18	27	14	137
Number found infected.....	3		1	3		2	4	1	14
Percentage infected.....	8		4	23		11	15	7	10

This table shows that of the 1,096 men examined 337, or 31 per cent, were infected with hookworms. In this table the mine employees are considered as in two classes, underground workers and surface men. It will be noted that of the 969 underground workers examined 363, or 38 per cent, were infected; whereas of the 137 surface workers examined only 14 men, or 10 per cent, were infected; moreover, all of the 14 infected men had formerly worked underground.

NONINFECTION AMONG SURFACE WORKERS.

The results shown in Table 1 indicate that hookworm infection is closely related to the mining industry, for if only 10 per cent of the surface workers at mines were infected, it is reasonable to expect that in this particular district few workers not miners are infected. It is somewhat surprising that in a mining district where 337 miners daily evacuating hookworm eggs the infection had not become widespread among men engaged in other occupations. This absence of infection among the nonmining inhabitants can be explained in

two ways. First, the number of unprotected, insanitary privies in the towns where the miners live is not large, and the privies that are present have earth vaults or are placed over a stream, so that the infectious bowel discharges are not spread about and absorbed by men. Many of the houses have sewers which empty into the creeks. The second protective feature is the long dry season in the district, which would tend to destroy and make nonviable a large number of worm eggs that were voided indiscriminately. The total annual rainfall for this district during 1915 was 27.69 inches and for the months of June, July, August, September, and October was absolutely dry, no precipitation whatever being recorded.

INFECTION AMONG MINERS.

The fact that such a high rate of infection prevails among miners in the Mother Lode mines would seem to prove offhand that the infection is contracted in these mines, but the labor is of a transient type; miners are constantly coming and going. The average duration of employment of the 969 underground men examined was 12 years. Hookworms will live inside of a man for more than 10 years; therefore the fact that miners in this district were found infected does not necessarily prove that the infection was contracted in these Mother Lode mines. The miners may have been infected before they came to these mines.

The evidence, however, goes to show that the mines of the Mother Lode are infected and that all the infection is not imported. An important point is brought out by the following facts:

1. The low percentage of infection among surface employees compared to the high percentage of infected underground miners indicates indirectly that the mines are infected, because if the infection was all imported the proportion of infected men on the surface would probably be larger.

2. In one mine examined, 75 of the underground men were found to be infected and 75 were not. It was found that the duration of previous mining experience of the infected men was about 12 years, as that of the noninfected men, averaging $5\frac{1}{2}$ years for the infected men and 5 years for the other men. If the infection was imported, that is, if it depended upon their previous experience rather than their present work, it would appear that the infected men would have had a much greater previous experience than the noninfected men.

3. The date of employment of every man examined was noted, and the duration of his service at the present mine up to the time of his examination was figured. It was found that the average duration of employment of the infected men was 12 years and that of the noninfected men was 3 years. As the noninfected men had not

these mines so long as the infected men had, it would appear that there is a relation between the mines and the source of infection.

At one mine it was found that 17 of the 77 infected men had never worked in mines elsewhere, and that 4 men were infected who had previously taken the hookworm treatment and had not worked in any other mine since they had taken the treatment. This indicates that this particular mine is infected and the similarity of conditions between this mine and other Mother Lode mines is presumptive proof that the mines are infected.

SOURCE OF INFECTION NOT KNOWN.

Hookworm could have been introduced into these mines from any one or all of several different countries. The infection may have been imported by miners from Cornwall, England, as there are many Englishmen in the mines of the Mother Lode and it has been proved that Cornwall mines were once infected, or it may have been introduced by Italians, Spaniards, or Mexicans, as many of the miners are of such extraction, and the Rockefeller Hookworm Commission has found that all of these countries are infected. It is not important to know where the infection came from, although to determine the sources of hookworm would be of assistance in deducing their origin. An important fact to be faced is that hookworm infection is present in these mines and ought to be eradicated.

PRESENT UNDERGROUND TOILET FACILITIES.

There are many contributory factors to the spread of hookworm infection, but all of them operate only after the infectious eggs have been set at large by the careless disposition of human excrement. The source of all hookworm infection is in the discharges from the humans. Information in regard to the underground toilet accommodations at the mines examined is shown in Table 2 following.

TABLE 2.—Data in regard to underground toilet facilities at mines examined.

of mine.....	A.	B.	C.	D.	E.	F.	G.	H.
accommodations, underground, proposed to go to surface.....	Yes.....	Yes.....	Yes.....	No.....	No.....	Yes.....	Yes.....	Yes.
proposed to use pump.....	No.....	No.....	No.....	No.....	Yes.....	No.....	No.....	No.
are cans placed in accommodations.....	No.....	No.....	No.....	Yes.....	No.....	No.....	No.....	No.
number of years in use.....	Yes.....	Yes.....	Yes.....	No.....	No.....	Yes.....	Yes.....	Yes.
number of times per year cleaned.....	5.....	1.....	1.....			2.....	1½.....	1.
.....	2.....	2.....	7.....			1.....	1.....	1.
date of inspection of (1916).....	{Feb. 17, 19, 21, 23, 25.	{Mar. 20, 21, 22.	{Apr. 19	{Mar. 20 June 26	{May 9	May 5	May 13	May 22

Six of the eight mines examined had toilet cans, which were supposed to be used by the men and which were brought to the surface and emptied at regular intervals. At one of the mines the men were supposed to go to the surface when necessary; at another mine the men defecated in an inconvenient and insanitary way into the sump. With the exception of one mine the introduction of toilet cans is of comparatively recent origin, and the delay in introducing their use has doubtless contributed largely to the spread of hookworm infection.

IMPROVEMENTS IN UNDERGROUND TOILET FACILITIES RECOMMENDED

Although the basic features of the present underground toilet systems are correct in that the infective material is collected and removed from the mine, certain details should be improved. Not only should the toilets be installed, but their use should be strictly enforced, and to require this the toilets must be maintained in a usable condition. They should be emptied at least thrice a week or, preferably, every night, as is done in one mine. The cans that were emptied once a week had an odor so vile that it nauseated a person to use them. If the can is situated in a place that makes water, galvanized iron sheets should be placed over the can like a shed roof, to prevent the seat becoming wet from drippings.

It is believed that if the cans are one-quarter filled with water with some disinfectant added at the time they are placed in the mine, the conditions would be better than if a powdered disinfectant is used. Under any circumstances the work of cleaning these cans is a disagreeable task, but if the contents are maintained in a liquid condition by the addition of water the deposits will not adhere to the sides or corners of the can, but will readily pour out, and the churning of the contents, odors, and ineffective cleaning, incident to the emptying of the cans, will be avoided. If water is to be added to the cans, they should have a tight cover and one that is readily removable must be provided for each of the cans, so that the material will not splash out while the can is being conveyed to the surface.

Some of the mine superintendents complained that the toilet cans were abused and defiled by some of the miners, the trouble being caused by the men standing instead of sitting upon the seats. Two measures might prompt the men to squat upon the seats—fear of contracting hookworm or disease or because the seat is wet or defiled. These reasons are not rational ones, and a properly designed toilet will do much to correct the abuses.

* For discussion of sanitation in mines see White, Joseph, *Underground Sanitation in Mines*: Tech. Paper 132, Bureau of Mines, 1916, 23 pp.

The design of an underground toilet in which careful attention has been given to details is shown in figures 1 and 2. Note the sanitary seat open back and front.

Plate I shows three views of an underground toilet invented and sold by a mine-supply house in California. An ingenious feature is the way in which the disinfectant is automatically thrown into the toilet after each use.

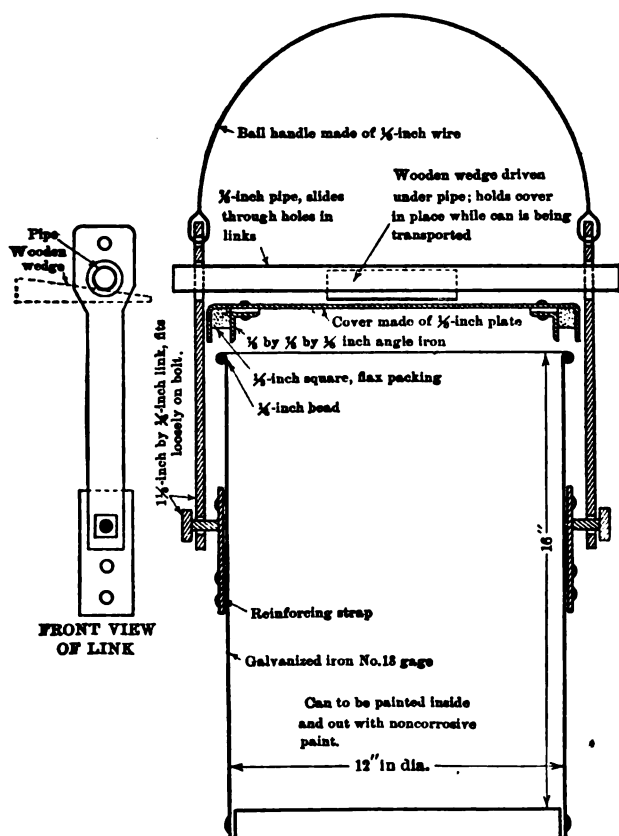


FIGURE 1.—Portable privy can for use in mines.

EMPTYING CANS AT SURFACE.

The method in which the contents of the cans should be disposed of when they are brought to the surface depends largely on local conditions at and around the mines. The material must be so disposed of that it will not pollute a drinking-water supply, so that it will not be disseminated by flies, and so that no nuisance shall be created. In some places it would be satisfactory to empty the cans into a covered leaching pit into which lime is sprinkled at regular intervals. This pit should be so situated or protected as to

prevent pollution of the water supply. At some mines this could be passed through a septic tank of approved design. In a successfully operated septic tank the solid matter in

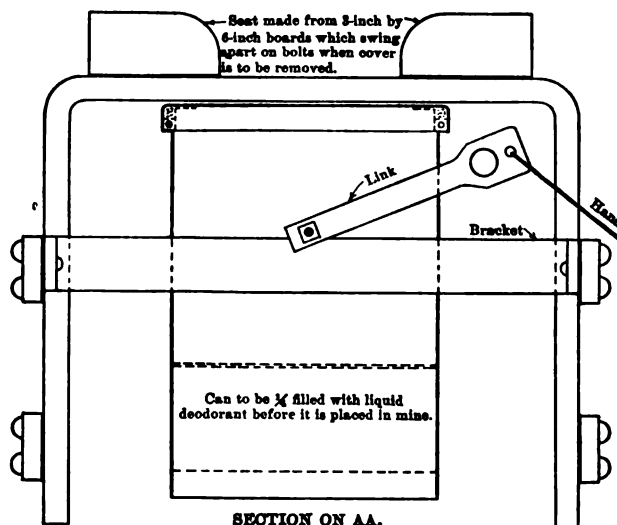
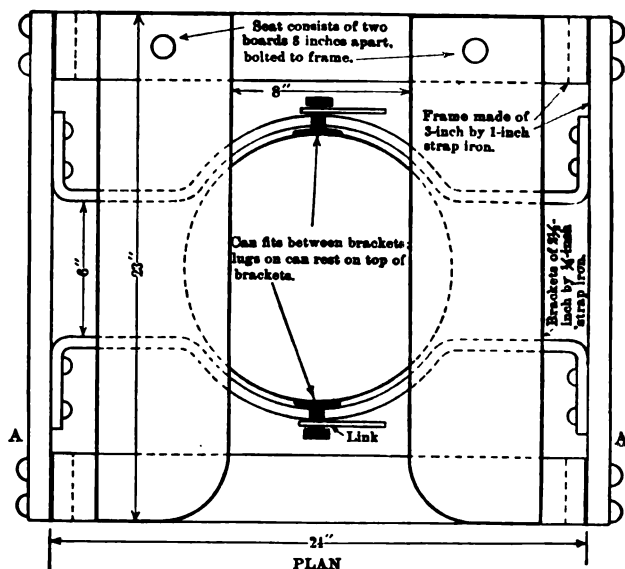
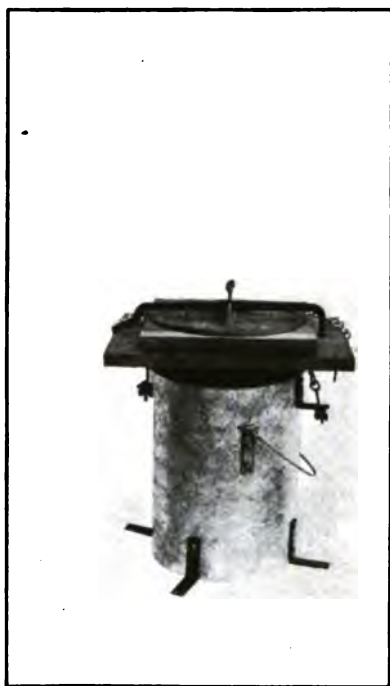


FIGURE 2.—Portable privy can placed in commode.

decomposed and liquefied and many of the disease-causing are destroyed. The effluent from a septic tank, however, may require additional treatment if it flows into a drinking-water supply.



A. PRIVY CAN, READY FOR USE.



B. PRIVY CAN, CLOSED FOR REMOVAL.



C. PRIVY CAN, WITH COVER REMOVED FOR CLEANING.



septic tank is used, strong disinfectants should not be added to the contents, as this would kill the bacteria upon which the septic tank depends for its liquefying function.

SURFACE TOILETS.

It is believed that if a modern, sanitary, water-flush closet were installed on the surface near the change house many of the miners would use this instead of waiting until they went underground. The time of bowel movement can be regulated to a large extent. One of the recommendations made to the miners is that they use the underground toilets only when necessary. Most of the present surface closets are foul smelling, inadequate, and inconveniently located.

An inexpensive water-flush closet is shown in figure 3. The price of a closet similar to the one shown but with four sections instead of two is about \$175 f. o. b. San Francisco, Cal. It is built in sections and any number of seats can be ordered. It flushes automatically at any interval desired, and it does not get out of order easily. The vent from this can be conducted to the same pit or disposal tank in which the mine cans are emptied.

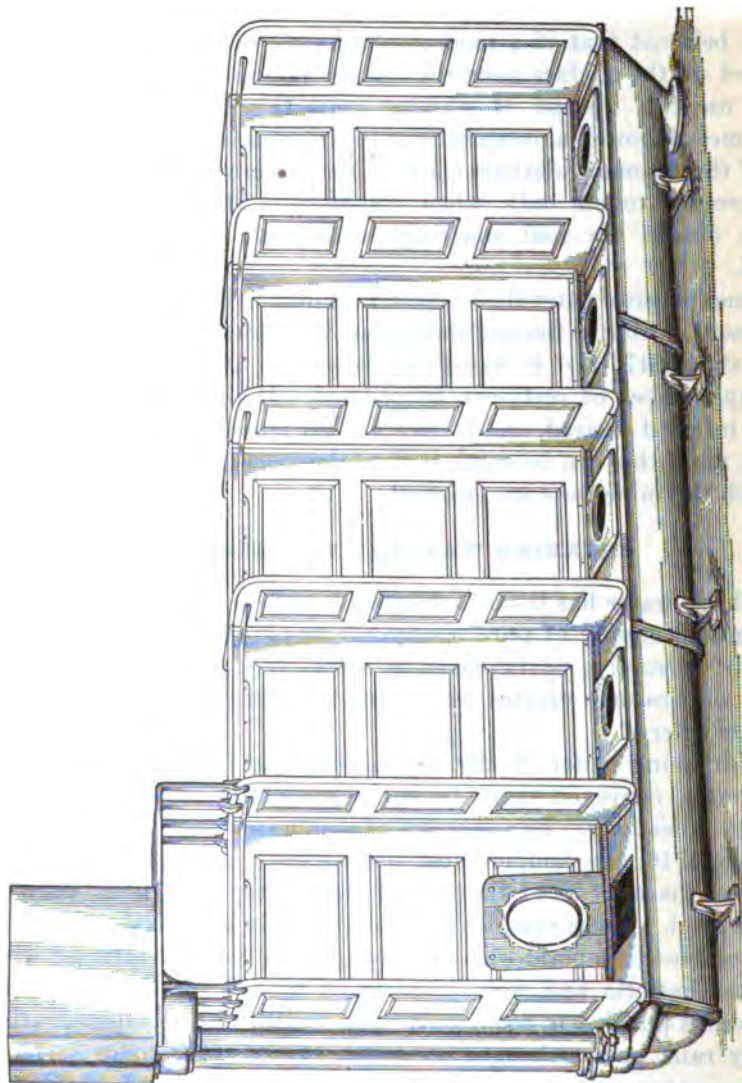
DRINKING WATER IN THE MINES.

Drinking water has little to do with the spread of hookworm, but an abundant supply of pure wholesome drinking water is so fundamentally necessary, particularly in mines, where working in such high temperatures creates an abnormal thirst, that a few suggestions are offered.

The drinking water chiefly in use over the district examined is surface water, conveyed there from the mountains in an open ditch about 40 miles long. Two samples of this water were taken on February 28, 1916, one sample being drawn from the tap at the collar of one of the mines and the second from a tap in a drug store in Jackson, Cal. These samples were analyzed in the California State Board of Health laboratories and found to be heavily infected with intestinal bacteria and were pronounced unsafe for drinking purposes unless purified in some way. The samples were collected after heavy rain, and, although, no doubt, they showed more bacteria than the water contains normally, it is important that a drinking-water supply be judged by its worst, not its best, conditions. The sanitary chain is no stronger than its weakest link.

All of the mines except one either clarified the water by passing it through small filters attached to the water pipe or depended on natural filtration through the ground. No test was made of the efficiency of these filtration methods. One mine received its drink-

ing-water supply from a spring located near by. During the summer all of the mines add ice to the drinking water, and one operates a refrigerating plant for this purpose. One mine, deep one, has bubbling fountains at the mine collar and at the stations underground.



In seven of the eight mines examined the drinking water was carried into the mines in kegs, varying in capacity from 3 to 5 gallons. These kegs were placed at the underground stations and at the crosscuts or drifts convenient to the workingmen. Fresh water was taken into the mines several times a day.

The manner in which the men drink from these kegs is seriously hygienic and should be corrected at once. In some of the kegs a small iron nipple is screwed into the end, and the men's lips come in contact with this when they drink. In some of the mines there is only a hole in the side of the keg and the men place their lips to the hole and tip the keg. This procedure not only permits the lips to come in contact with the keg, but it also permits the water to siphon through the drinker's mouth and back into the keg. The writer examined the death certificates of 291 miners of Amador County. On 96 of these, or 33 per cent, tuberculosis was assigned by the attending physicians as the cause of death. It is well known that tuberculosis is a lingering disease—the average age of the 96 men deceased from tuberculosis was 49 years—and that tubercular victims often continue to work when the disease is in its advanced infectious stages. The hole in the keg is a common drinking place for miners suffering from tuberculosis, syphilis, or other diseases, and miners in good health. Another objectionable feature is that men drinking tobacco may drink from the hole. A simple and inexpensive way of preventing the lips coming in contact with the keg is shown in figures 4 and 5.

The water kegs should be steamed out thoroughly every night. At some mines the superintendent's orders regarding care of water kegs are not always carried out. At the mine a sample of water was taken from the tap at the collar where the kegs are stored and a sample was drawn from one of the kegs in the mine. The keg water sample contained more bacteria than the sample drawn from the tap.

EARLIER INSANITARY CONDITIONS.

Before the introduction of toilet cans into the mines the workmen defecated either in filled stopes or in abandoned crosscuts, in drainage sumps, or in mine cars or ore bins. The infected feces in the course of time became spread throughout the mine, either being tracked about on the miners' shoes or spread by mine cars or possibly by rats. The writer believes, however, that the mine water was an important disseminating agent.

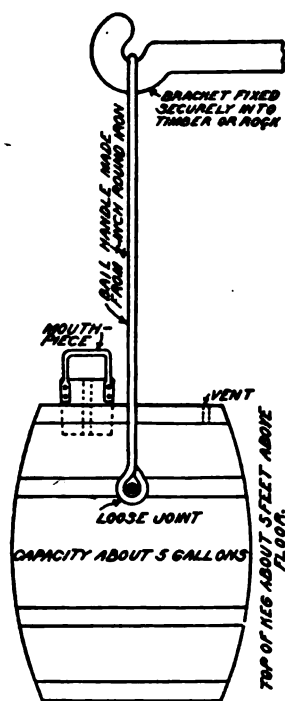


FIGURE 4.—Drinking-water keg for use in mines.

VIABILITY OF HOOKWORM EGGS.

It has been stated that hookworm eggs will remain infectious only about a year after they have been evacuated from the intestine. This may be the case when the eggs are exposed to the variable climatic conditions on the surface; but hookworm eggs apparently remain infectious for a longer period than one year when they are

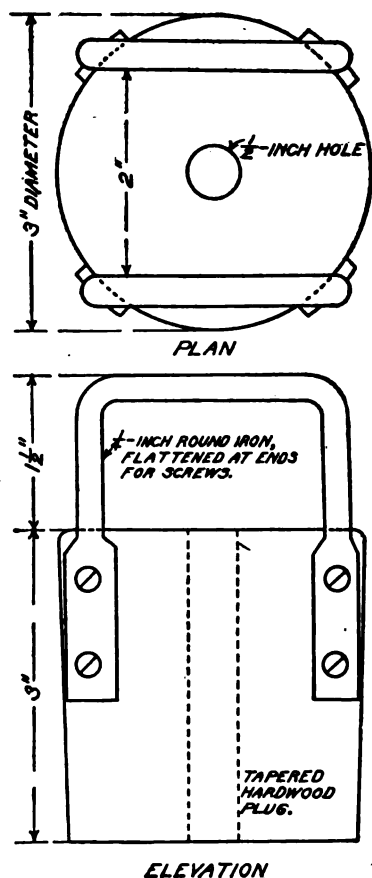


FIGURE 5.—Sanitary mouthpiece for drinking-water keg.

larvæ can bore through the intact skin at whatever place infection happens to come in contact with the body. A miner can not work without getting dirt on his hands and clothing; therefore safety lies in keeping the dirt of the mine as noninfectious as possible. With a mine once infected, the means whereby the mine becomes infected are obvious. They are discussed in detail on the following pages.

deposited in the mines, where the temperature conditions remain uniform year in and year out. At the mines, at which 60 per cent of the underground men are infected with hookworm, installed dry about five years ago, and the law has been enforced to some extent in spite of these sanitary precautions. Eight men were found infected who never worked in mines elsewhere and entered this mine after the installation of the closets. A few men were treated for hookworm since the installation of the closets were to be reinfected. This leads me to believe—although it is doubtful—that possibly some of the infections results from the survival of infections in places created five years ago in the mine before the cans were installed.

WAYS IN WHICH HOOKWORM IS CONTRACTED AND DISSEMINATED.

Many kinds of germs can enter the body through the mouth or through the skin when the skin has been cut or broken. In addition to these two ways, hookworm

MINER'S SHOES.

hookworm-infected countries where the inhabitants go without shoes, the logical place for hookworm larvæ to gain entrance to the body is through the feet. Hookworm larvæ might easily work their way through a fairly good pair of shoes, but the infection would be light. When a mine is known to be infected, a miner should take unusual care to keep his shoes in good repair. At a change house of one infected mine 107 pairs of miners' shoes were examined and 43 pairs, or 40 per cent, were found to be broken down so as to permit hookworm infection. The miner's shoes are a large item of expense. Mine shoes cost about \$5 a pair, and in some mines they last only four months. Considering the repairing necessary, the average miner will spend about \$20 per year on shoes. Mining companies could save the miner some expense if shoes were kept in a storehouse and sold to the miners at cost.

MINE LADDERS.

Some of the underground workmen may contract hookworm from mine ladders. The infectious dirt gets stuck on the shoes and it is scraped off the shoes onto the rungs of the ladders, whence it rubs against the skin of the hands. Figures are given on page 31 which show that bosses and repair men, who move about the mines extensively and frequently climb ladders, are more apt to become infected than men who work steadily in the stopes.

RATS.

Rats possibly spread the infectious material. In six of the eight mines examined rats were observed. At one mine three rats were trapped and their intestines examined for hookworms and then their feces examined for hookworm eggs. None was found. Although hookworms are not known to occur in rats, they may, however, spread the hookworm eggs mechanically by their legs and droppings. The workman who empties the toilet cans at one of the mines stated that rats often get into the toilet cans. At some of the mines large numbers of rats are present and are tame, being fed and "petted" by the miners. At lunch time, when the miners gather at the station to eat, the rats likewise assemble and the miners throw them scraps of food. Some of the miners believe that the rats warn them of unseen dangers, and say that the rats can tell by instinct when the roof is unsafe. The high proportion of accidents due to falls of rock makes one doubt the truth of this statement, and it is doubtful whether rat-infested mines have more accidents than mines without rats. At any rate, these disease-

carrying pests should be eliminated from mines regardless of superstitions miners may have regarding their usefulness.

UNDERGROUND EATING STATIONS.

Hookworm larvæ may gain entrance to the body through the mouth, and the conditions under which the miners eat make possible a source of infection. Facts regarding this practice are given in Table 3.

TABLE 3.—Data showing toilet facilities available during underground work.

Name of mine.....	A.	B.	C.	D.	E.	F.
Do miners eat underground.....	Yes..	Yes..	Yes..	Yes..	No...	No...
Facilities present for washing hands.....	No...	No...	No...	No...	Yes..	Yes..
Benches present at underground stations.....	No...	Yes..	No...	No...	No...	No...
Garbage cans present at underground stations.....	No...	No...	Yes..	No...	No...	No...

Conveniences for eating in the mines are, in most cases, unsatisfactory. Shortly before dinner time the dinner buckets are sent down from the surface. The miners assemble at the station, sit on old timbers or on the floor, their buckets being held between their legs. It is said that sometimes food is spoiled by the buckets getting tipped over. At some mines fresh drinking water is sent down at meal times, but tea or coffee is the favorite beverage. Some of the miners wash their hands in water poured from a drinking-water keg; others are said to wash their hands in a drainage ditch; many eat unwashed. Apparently the only reason for perpetuating these conditions is that such has been the practice at these mines for years past and so it goes on. This statement applies to most mines in the United States. Mining, at its best, is not a pleasant occupation, and has many disagreeable features that are difficult to correct, but this is not the case with eating underground. In the periods of the day when the underground mine station is used for its numerous other functions the underground mine station becomes the miners' dining room; why not make it as such? Water for washing the hands should be provided at benches, allowing about 2½ feet of seating space per man, and benches should be placed in the stations for the men to sit on while eating; garbage should be provided for food remnants and their use enforced; the station should be well lighted and kept as clean and as free from unnecessary débris as possible.

MINERS' WASH AND CHANGE HOUSES.

All of the mines on the Mother Lode have had wash and change houses for a number of years, although these are not required.

no charge is exacted from the men for their use. There are many mining districts in the United States where this improvement is just being introduced, and many mines are still without them. Information about the wash and change houses in the Mother Lode district is shown in Table 4.

TABLE 4.—*Information on wash and change houses at mines examined.*

of mine.....	A.	B.	C.	D.	E.	F.	G.	H.
Has a wash and change house?.....	Yes.	Yes.	Yes.	Yes.	Yes.	Yes.	Yes.	Yes.
Has concrete floor?.....	No.	No.	No.	No.	Yes.	No.	No.	No.
Is it neat and clean?.....	Yes.	Yes.	No.	No.	No.	Yes.	No.	No.
Number of showers present.....	13	6	10		1		2	2
Facilities for washing hands satisfactory?.....	No.	No.	No.	No.	No.	No.	No.	No.
Are common wash basins in use?.....	Yes.	Yes.	Yes.	Yes.	Yes.	Yes.	Yes.	Yes.
Facilities for boiling underground working clothes discarded?.....	No.	No.	No.	No.	No.	No.	No.	No.
Are face toilets satisfactory?.....	No.	No.	No.	No.	No.	No.	No.	No.

Many of these houses have been in use for some years and do not possess the up-to-date conveniences of the more modern wash and change houses. Improvements have been developed in the construction of washhouses as in every other branch of mining. It would be unreasonable to expect these washhouses to be abandoned, but inexpensive improvements should be made on them. There is no excuse for the objectionable condition in which some of the washhouses are found. When a new floor is required the wooden floor should be replaced with a well-drained concrete floor, and it would be well to have concrete extend up the sides for a few feet. Showers should be installed at most of the washhouses. The facilities for washing the hands and face should be improved. At present metal washbasins are used, and some miners bend over the tub and wash only the upper part of the body. This practice is common among only certain classes of miners. Provision should be made for the men to wash their hands and face in flowing water, and the common washbasins should be discarded.

FACILITIES FOR WASHING THE MINERS' WORKING CLOTHES.

Facilities should be provided in all washhouses so that the miners can boil their underground working clothes. It has been claimed, with some truth, that the miners' underclothes are washed less frequently since the introduction of washhouses than they were previously, as when the miner wore his "shift" clothes home his family would see that they were washed, but when the miner leaves his working clothes at the mine this supervision is missing. Towels should be boiled often, at least three times a week. If a number of galvanized iron tanks about 20 inches in diameter and 24 inches deep are installed and connected with hot-water or possibly live-steam

lines the condition in which the working clothes and towels kept would no doubt be much improved.

MINE-DRAINAGE SYSTEM.

It is believed that mine drainage has been and is an important factor in the spread of hookworm infection, as the mine water, the humidity that appears to be essential to the development of hookworm eggs, and furnishes a medium for conveying them all over the mine. If the mine made no water the infection would remain where the stools were evacuated and few miners would come in contact with it. If, however, an infected miner commits a nuisance in a wet stope the mine water filtering through the stope picks up the hookworm eggs and conveys them to the traversed levels, or if the miner commits a nuisance in an abandoned crosscut which makes water the infectious material, it is re-conveyed into the active parts of the mine.

In an upper abandoned level of one of the mines examined, a stool was found partly submerged in a drainage ditch. A specimen of this stool was microscopically examined and it was found to be infected with hookworm eggs. Although eggs that are under water do not develop, the eggs and the larvæ are doubtless scattered all over the water. This furnishes a concrete example of how hookworm infected material may be spread in a mine.

It is not the quantity so much as the way in which the mine water is taken care of that is important. In mine B, which uses 125,000 gallons daily, only 16 per cent of the men were infected. Possibly more attention was given to the drainage system in this mine than in any of the other mines examined. An effort was made to intercept the water at each level and not permit it to flow through the stopes. The intercepted water was conveyed to sumps and maintained ditches to sumps at the stations. On some of the upper levels the water was carried in small wooden boxes or flumes, and this helped greatly in keeping the passageway and footboards dry. The foreman at this mine said it pays to keep the mine water out of the stopes, because, besides making the stopes mucky, disagreeable and dangerous to work in, the water saturated the "gouge" for timbering and caused the ground to swell and thereby increased the expense of pumping was less if it was intercepted on the upper than the lower levels.

In mine C, which made 190,000 gallons of water, 67 per cent of the men were infected. The mine water was not carefully kept out of the mine ditches not being well maintained, and in several places the footboards were flooded and mucky.

CHARACTER OF MINE WATER IMPORTANT.

said that hookworm larvæ will not live in mine water which has 2 per cent of salt^a or if the mine water is 0.2 per cent acid saline.

Medical confirmation of this has been found in the conditions exist at the Levant mine, near St. Just in Cornwall. Tin, copper, and arsenic are extracted from this mine. Although it is one of the very deep mines of Cornwall, it is also one of the most healthful. In this mine the eggs of many of the intestinal parasites that in other mines were found, but in the fecal matter lying about no eggs of hookworm were detected. Their absence was attributed to the fact that as the mine is under the sea the sea water filters through the mine into the mine.

Some of the inland coal mines of France, especially those between Anzin and Denain, Manouvriez^b found that the waters percolating into the mines were saline, and that they probably came from subterranean caverns, remnants of ancient seas or of salt-water lakes above the carboniferous strata. Analyses showed that these waters contain 0.7 to 2 per cent of common salt. It is interesting to note that in those pits in which the water contains the higher percentage of salt there are no cases of hookworm infection among the miners, although the malady is present in other pits not so fortunately watered.

The water in the mines of the Mother Lode is alkaline. A sample of the water from one of the lower levels of one of the deepest mines was analyzed and was found to be only 0.043 per cent alkaline, whereas to be an effective sterilizing agent it should be about 0.2 per cent alkaline. A specimen of feces found partly submerged in a drainage ditch of one of the mines, was collected, care being taken to select the specimen from the submerged part of the stool. The specimen was found heavily infected with hookworm eggs.

Specimens of the excreta from 204 miners who work in one of the mines in the northern part of the State of California were analyzed. The water in this mine is highly acid, analyzing 0.01 per cent acid. Only 3 per cent of the men examined were infected, and the possibility that these men brought the infection with them rather than contracting it at the mine in question, must be considered.

CONGESTION IN THE MINES.

One of the reasons why hookworm infection became widespread in the mines of the Mother Lode, is due perhaps to the steep dip of the

Manouvriez, A., Prophylaxis of ankylostomiasis: Brit. Med. Jour., vol. 161, pt. 2, 1905, p. 1418.

Manouvriez, A., article cited.

ore deposit, averaging as it does about 70° to 80°. As the developed one below the other, the mine water which drains the old upper infected levels leaks down into the new work is as if in a 20-story building, the tenants on the top floor their waste water to leak through upon the residents on the tenth floor, and then those on the eighteenth floor, after the waste from the nineteenth and twentieth, would add to it and pass it on to the tenants on the seventeenth, and so on. If cans had been installed in the mines of the Mother Lode 2 years ago, rather than 2 years ago, hookworm infection might not have gotten a foothold.

In most of the coal mines in the United States to-day there is absolutely no toilet provision whatever and the miners carry their excrements all over the mine. The coal deposits, for the most part, are in a horizontal strata, and their development spreads over great areas rather than sinking to great depths. This perhaps helps to explain the fact for the fact that coal mines are not heavily infected with hookworm. The toilet accommodations in coal mines are, as a general rule, better than in the metal mines.

The approximate slope of the shaft, the number of miles developed, number of active working levels, depth of lowest level along the slope, approximate amount of water removed daily, approximate number employed underground, and percentage of underground men found infected are shown in Table 5.

TABLE 5.—Data on slope, development, drainage, and proportion of underground workers infected with hookworm in the mines examined.

Name of mine.....	A.	B.	C.	D.	E.	F.
Approximate slope of shaft, degrees..	60	90	70	70	50	50
Number of mine levels developed.....	35	16	24	6	5	5
Number of active working levels.....	5	13	6	5	4	3
Depth of lowest level (along slope), ft.	4,200	3,750	2,850	3,200	1,400	700
Approximate water removed daily, gallons.....	40,000	125,000	190,000	20,000	145,000	30,000
Approximate number of men employed underground.....	135	270	260	60	55	30
Percentage of underground men found infected.....	60	16	67	39	23	21

TEMPERATURE OF THE MINES.

A number of temperature and humidity readings were taken at various points in the mines. In selecting the places for the readings an attempt was made to get a fair average temperature of the active part of the mine. Some temperature readings were taken in abandoned levels, but these were not included in computing averages. Information as to temperature and relative humidity is shown in Table 6:

3.—*Temperature and humidity of mines examined, and number of men infected with hookworm.*

Mine.....	A.	B.	C.	D.	E.	F.	G.	H.
(temperature and humidity taken.....	32	31	24	18	11	21	32	16
depth in feet) between which were taken.....	3,600 and 4,200	2,000 and 3,750	1,800 and 2,850	700 and 3,200	900 and 1,400	500 and 700	1,200 and 2,400	1,650 and 2,350
temperature observed, °F.....	88	84	83	79	70½	68	79½	77
temperature observed, °F.....	80	67	70	61½	62	58	72	67½
temperature, °F.....	83	78	75	73	68	63	76	72
relative humidity.....	100	100	100	100	100	100	100	100
relative humidity.....	92	91	95	94	93	95	96	92
relative humidity.....	97	97	99	97	98	99	99	97
lowest working level along.....	4,200	3,750	2,850	3,200	1,400	700	2,400	2,350
of underground men in.....	60	16	67	39	23	21	33	31

Table shows that the highest temperature found in any of the was 88° F. and the lowest was 58° F. Mine A had the highest temperature, 83° F., and mine F had the lowest average temperature, 63° F.

Thomas Oliver, in an article entitled "Miners' Worm Disease," April, 1905, issue of the *Journal of State Medicine*, states that temperatures between 68° and 90° F., with a certain degree of moisture favorable circumstances for the development of hookworm into larvæ. All of the mines examined on the Mother Lode, one, have an average temperature between these limits. The of the hookworm infection, however, does not vary with the temperature, as mine B, which is second highest in temperature, least infected of all the mines. The author believes that the infection once introduced, it is not so much the warmth of mines as the uniform temperature accompanied by high humidity makes the deep mines of the Mother Lode favorable for the development of hookworm infection. Table 6 shows an interesting relationship between depth and temperature.

With the exception of mine A all of the mines were connected at least two shafts so that natural ventilation existed. Mine A is still pushing to completion its connection with a second opening. Besides the depth and number of openings, a factor influencing the temperature is the amount of timber in the mine. Some of the hot-temperatures were found to be near stopes which had been filled with old mine timbers. Most of the mine superintendents now believe that the practice of filling stopes with old timber is false economy. Besides increasing the temperature and thereby reducing the efficiency of the workmen, the fire hazard of the mine is increased. In most of the mines the old timber is now hauled to the surface and burned under the boilers.

RELATIVE HUMIDITY.

The high relative humidities of the mines examined, Table 6, help to make these mines fertile breeding places for the hookworm. The average temperature in the mines may be affected by the temperature outside, but the relative humidity of the mine air appears to have little relationship with the relative humidity of the intake air. Temperature and humidity readings were taken at the surface at mines D, F, and H on the day of their examination. The difference in humidities between the surface and the underground is far more striking than the difference in temperature, as shown in Table 7.

TABLE 7.—*Relative temperature and humidity in mines examined.*

Name of mine.....	D.	
Date of examination (1916).....	June 26	Ma
Temperature at surface, °F.....	67½	
Average temperature underground, °F.....	73	
Relative humidity at surface.....	59½	
Average relative humidity underground.....	97	

Attention is directed to Table 8, which shows the annual average temperature and surface temperature at Jackson, Cal. It will be noted that in 1915 the average temperature on the surface is 59.4° F.; the lowest is 45.4° F. and the highest is 76.4° F. The variable conditions of the surface are a contributing factor in the lower percentage of hookworm infection among the surface workers.

TABLE 8.—*Mean temperature and precipitation at Jackson, Cal.*

Month (1915).	Mean temperature, °F.	Precipitation (inches).	Month (1915).	Mean temperature, °F.
January.....	45.4	7.53	August.....	76.4
February.....	46.5	7.63	September.....	67.7
March.....	55.0	1.24	October.....	64.4
April.....	57.0	2.27	November.....	53.3
May.....	58.4	0.93	December.....	46.5
June.....	68.0	0	Annual.....	59.4
July.....	74.9	0		

* Readings were taken at Kennedy mine at Jackson, Cal. Results from annual climatic observations at California section, Weather Bureau, U. S. Dept. Agriculture.

RELATIVE INFECTIOUSNESS OF THE VARIOUS MINES EXAMINED.

The percentage of infection found among the underground employees does not necessarily measure the relative extent of the infection in the mines. Attention is called to this fact that the mines are infected with hookworm eggs; the miners are infected with hookworms. These will be differentiated hereafter by the results to the infection among the miners and the infectiousness of the mines.

The length of time that the men have worked in the mines perhaps influences the percentage of infection found. It is

that the likelihood of contracting hookworm infection depends upon the period that the men have been exposed to infectiousness. No doubt some of the miners having hookworm contracted infection in a different mine from the one in which they were working at the time they were examined, but as this would apply to all the mines these cases would perhaps be balanced in attempting to determine the relative infectiousness of the various mines examined. If all the mines were equally infectious, other factors being the same, there should be a relation between the percentage of infection and the average duration of employment. The mine with the highest percentage of men infected should have the longest average duration of employment. Table 9 shows this not to be so. Mine C, with the highest percentage of infection, had the second shortest period of employment. This indicates that the extent of infectiousness in the different mines varies.

If we take into consideration this time element in estimating the relative infectiousness of the mines, it might be assumed that if at mine A 60 per cent of the men were infected in 48 months, 100 per cent would be infected in 80 months, and that in mine B, if 16 per cent were infected in 26 months, 100 per cent would be infected in 162 months. Comparing 162 with 80, it might be said that mine A was twice as infectious as mine B. This is not actually so, but is believed to be nearer the truth than if, on comparing 60 per cent with 16 per cent, it was said that mine A was 3.8 times as infectious as mine B.

Table 9 shows the percentage of infection, average duration of employment, estimated time required for 100 per cent infection, and the relative infectiousness of mines based on the percentage of miners infected, and the relative infectiousness based on the infectiousness of the mine, for the various mines examined. In comparing these two ratings, it will be observed that mine A which is second worst according to the first method becomes sixth worst according to the second method. Mine E which is fourth worst according to the first method becomes third according to the second. The other mines occupy about the same relative standing with both methods.

TABLE 9.—Percentage of men infected with hookworm and duration of employment in mines examined.

Mine.....	A.	B.	C.	D.	E.	F.	G.	H.
Percentage of underground men found infected.....	60	16	67	39	23	21	33	31
Average duration of employment, months.....	48	26	13	13	8	19	23	17
Estimated time required for 100 per cent infection, months.....	80	162	19	33	35	91	70	55
Relative infectiousness based upon percentage of infected men.....	2	8	1	3	6	7	4	5
Relative infectiousness based upon estimated time required for 100 per cent infection.....	6	8	1	2	3	7	5	4

IMPERMANENCY OF LABOR IN THE MOTHER LODE MINES

Hookworm infection no doubt is prevalent in other mines as well as California. The desirability of extending this work to other mining States became apparent at the very outset of the investigation. The labor on the Mother Lode is in a constant state of flux. Numbers of miners were examined and found infected with hookworm, but before treatment could be administered they had gone. Their whereabouts is not known, but in whatever mine they work they will spread hookworm infection if the mine is infested and the conditions are "favorable" to the development of hookworm eggs.

Statistics which show the impermanency of labor in the Mother Lode mines are presented in Table 10. The average duration of employment of the underground men at each mine was determined. The date of employment of each man was looked up and the number of months that he had been employed was calculated. These figures were averaged in order to find the average duration of employment of the underground force at each mine. Anyone whose employment at any time took him into the mine was classified as an underground man. The longest average period of employment was 48 months at mine A; the shortest period was eight months, at mine E.

In addition to this the average number employed during the year was determined. The average number "paid off" each month during the year were found for some mines. In mine D there were more "turnovers" of the labor force during 1915. At mine C the men were paid off during 1915; the proportion of infected men was 67 per cent. If this rate is applied to the men employed about 241 men infected with hookworm left this one mine during the year 1915.

TABLE 10.—Data showing impermanency of labor in mines of the Mother Lode district.

Name of mine.....	A.	B.	C.	D.	E.	F.
Average duration of employment, months.....	48	26	13	13	8	19
Average number employed underground in 1915.....	136	270	250	62	57	19
Average number paid off each month during 1915.....		15	30		16	
Frequency of turnover of labor force in 1915.....		18	8.6		9	
Average service of Americans, months.....	61	35	23	17	9	13
Average service of Italians, months.....	47	23	4	13	8	19
Average service of Spaniards, months.....		18	7			
Average service of Austrians, months.....	44	32	18	10	7	2
Percentage of underground employees married.....	40				35	
Average service of married employees, months.....	55				10	
Average service of unmarried employees, months.....	40				7	

The lack of permanent mine labor has other serious results in addition to its effect upon hookworm conditions. Floating labor is not efficient labor. The newcomer must get acquainted with the working conditions, and in so doing he wastes not only his own time but the time of his partner, the shift boss, and the boss.

ing labor means uncertain labor. Often the mine is not open to capacity because of lack of men, and when working places and there are unoccupied, part of the investment is idle, whereas overhead expenses are about the same.

floating labor increases the risk of accidents with the consequent suffering, and disorganization. An output of a mine may be affected for many hours by a fatal accident, every miner more or less affected by the knowledge of the catastrophe. The greatest cause of mine accidents is falls of rock, and it is logical to think that the miner, having a knowledge of the rock, its strata, and peculiarities, can guard himself better against such accidents than a new man who is not familiar with his working place. Loyalty, the most intangible but most valuable of all assets, can be stimulated in floating labor. This perhaps is its most objectionable feature.

SUGGESTED REMEDIES.

It is believed that the labor supply is a serious problem in this district and must soon be faced earnestly by the mine operators, as by operators in many other mining districts in the United States. Though the writer does not presume to offer a solution, it is felt that this problem is studied with the same attention and zeal as are other mining problems, remedies will soon be forthcoming. For example, in some mines of the district the average duration of employment is 48 months, whereas at others the men leave in eight months. A recognition of the losses due to floating labor will be the first step toward its abatement.

Several suggested remedies are given here. More information about the men should be obtained at the time of their employment. Registration cards showing the name, nationality, age, marital state, dependents, previous mining experience, etc., of the applicant are now being used at some of the mines and they should be used at all mines. Such a system in itself tends to keep away undesirable men who come only with the intention of earning a few dollars and then roaming on. When a man is "paid off" a definite reason should be assigned for leaving; if he is discharged, the shift boss should report the reason; if he quits voluntarily, his reason for doing so should be ascertained if possible. Facts thus obtained might be the basis of remedies. At least it would place the employment question on a more systematic basis.

There appears, as is natural, a seasonal fluctuation; the men quit the mines during the summer to go to ranches, or into the lumbering business, or into the hop fields, or fruit orchards. Instead of this two weeks' respite being concentrated into the summer, would it not be better to spread it over the entire year by making every Sunday a

holiday throughout the district, only the presence of money is absolutely needed being permitted in the mines? If a miner "goes into the hole" seven days a week, week in and week out, it will not be long before he gets discontented and craves a change. The progressive superintendents in the district hold these views and are trying to eliminate Sunday work. The effectiveness of the Sunday holiday would work out better if the district unanimously adopted the plan. In some of the coal-mining districts fluctuating employment due to seasonal trade variations is one of the reasons why miners roam about. This does not apply to the metal mines of the Mother Lode district, as the mines can be worked throughout the year. Gold sells at a fixed price and many of the mines are owned by producers.

Living and social conditions in the towns adjacent to the mines exert an influence on the problem of labor supply. The towns adjacent to what might be considered the best town enjoy the best conditions for workers. The town conditions are a difficult situation to regulate, but it is the policy of many of the companies not to interest themselves in town affairs or to attempt to regulate the actions of the miners away from the mines. The relative advantages and disadvantages of the company-controlled mining town are too complex to discuss in this paper. Many of the good features of the mining town could be adopted with advantage, it is believed. Some of the mine superintendents encourage interest in outdoor sports such as baseball, by donations and liberal concessions in the working conditions of the players.

One of the mining companies recently presented a clubhouse for a town near-by for the benefit of the citizens in general, which would be of benefit to the miners.

A few of the companies exert more or less influence over the building and living houses maintained for unmarried miners, and a more thorough investigation into conditions at bunk houses might be profitable of beneficial results.

The possibility of starting a building and loan association to help kind and assisting the men to build homes is suggested by the fact that this method is being tried by operators in other districts. Unmarried men with families make steadier workmen than single men, it is well known. This statement is confirmed by the results shown in Table 10.

Schools exert an important influence. Special night classes could be formed for young men where practical subjects could be taught which would help to make them better miners. English classes for foreigners could be organized. The school system in the Mother Lode district at the time of this investigation appeared to be insufficiently correlated with the basic industry of the district.

d localization. Closer interrelation between the school system and the mining industry could no doubt be worked out to the advantage of both and of the community at large.

These few comments on the impermanency of mine labor are offered merely as suggestions and not as a solution of this complicated problem. If they are of any value whatever they will have fulfilled their purpose.

INFECTIOUSNESS OF DIFFERENT OCCUPATIONS IN MINE.

With the hope of learning which particular areas of the mine were infected, or if certain areas were more infected than others, the underground working force was divided into four classes according to the nature of their duties, and the percentage of infection among each was determined. The four classes were: (1) The bosses and repair men who travel all over the mine; (2) the skip tenders whose main duty is the shaft; (3) the miners who spend most of their time in the stopes; and (4) the muckers and car men who traverse the levels. The value of the results of this study is seriously affected by the variability of duties, the muckers of one day becoming miners the next, and the miners becoming repair men or bosses. The study is valuable as showing indications rather than absolute proofs.

The percentage of infection among the four classes of labor for each mine and the average duration of employment is shown in Table 11.

11.—Results showing percentage of employees infected with hookworm for different classes of labor, and duration of employment.

Kind of labor.	Number of men examined.	Per cent infected.	Average duration of employment, months.	Kind of labor.	Number of men examined.	Per cent infected.	Average duration of employment, months.
Bosses and repair men.	28	68	76	Mine E:			
Skip tenders.	6	83	91	Bosses and repair men.	9	22	7
Miners.	84	55	39	Skip tenders.	2	0	9
Muckers and car men.	40	38	29	Miners.	13	15	9
				Muckers and car men.	32	28	7
Bosses and repair men.	8	13	99	Mine F:			
Skip tenders.	3	67	65	Bosses and repair men.	5	20	22
Miners.	201	17	28	Skip tenders.	1	100	77
Muckers and car men.	80	14	14	Miners.	10	20	16
				Muckers and car men.	13	15	16
Bosses and repair men.	33	70	19	Mine G:			
Skip tenders.	2	100	38	Bosses and repair men.	31	45	46
Miners.	59	63	19	Skip tenders.	1	100	20
Muckers and car men.	76	34	6	Miners.	22	23	21
				Muckers and car men.	67	30	13
Bosses and repair men.	10	60	25	Mine H:			
Skip tenders.	2	50	12	Bosses and repair men.	16	31	20
Miners.	27	40	14	Skip tenders.	2	50	34
Muckers and car men.	17	24	12	Miners.	19	16	17
				Muckers and car men.	47	36	8

It will be noted that the skip tenders and the bosses and repair men show a higher percentage of infection than the miners and the muckers and car men.

No doubt this is partly due to the fact that the skip tender bosses and repair men have been in the mines for a longer time and the possibility of contact with the disease has been consequently greater. At the same time the nature of their duties, about as they do all over the mine, also makes the possibility of contact greater.

The high percentage of infection among skip tenders indicates that the shafts are very infectious. This is what might be expected, as all the mine water drains toward the shaft.

One would expect the car men to be more subject to infection than the miners, because they travel back and forth on the wet tracks every day, whereas the miners remain in the comparatively dry underground. The figures in Table 11 appear to contradict this statement, for in five of the eight mines the miners have a higher rate of infection than the muckers and car men. It should be kept in mind, however, that the miners may have worked as muckers or car men before they became miners, and they might have contracted the infection then. It should be noted that in three mines, although the duration of employment of the muckers is shorter than the miners, the rate of infection among the muckers is higher.

PROPORTION OF INFECTION AMONG DIFFERENT NATIONALITIES

The percentage of infection among the different nationalities in the separate mines and for all the mines combined and the duration of employment are shown in Table 12.

TABLE 12.—Results showing percentage of employees infected with hookworm and duration of employment, by nationalities.

Nationality.	Number of men examined.	Per cent infected.	Average duration of employment, months.	Nationality.	Number of men examined.	Per cent infected.
Mine A:				Mine F:		
American.....	26	50	61	American.....	7	100
Italian.....	14	21	47	Italian.....	8	100
Austrian.....	110	54	44	Austrian.....	12	100
Mine B:				Mine G:		
American.....	63	17	35	American.....	17	100
Italian.....	165	12	23	Italian.....	57	100
Spaniard.....	34	18	18	Spaniard.....	19	100
Austrian.....	25	44	32	Austrian.....	27	100
Mine C:				Mine H:		
American.....	17	35	23	American.....	7	100
Italian.....	18	44	4	Italian.....	39	100
Spaniard.....	61	36	7	Spaniard.....	32	100
Austrian.....	75	70	18	Austrian.....	7	100
Mine D:				Combined mines:		
American.....	12	42	17	American.....	168	100
Italian.....	29	41	13	Italian.....	350	100
Austrian.....	16	31	10	Spaniard.....	148	100
Mine E:				Austrian.....	288	100
American.....	19	21	9			
Italian.....	20	20	8			
Austrian.....	16	31	7			

combining the results from all the mines it will be noted that the Italians were least infected, the Americans second, the Spaniards third, and that the Austrians had the highest percentage of infection. Four of the eight mines examined the Italians were the lowest; in three of the eight mines the Austrians were the highest; and in four of the eight mines the Italians had a lower percentage of infection than the Austrians, although the average period of employment in the mine was longer than that of the Austrians. It would appear that this difference between Italians and Austrians is more than a mere coincidence, but a satisfactory explanation can not be given. It may be that the Italians look after their shoes better; it may be a question of personal hygiene; or it may be a dietary difference of the two nationalities.

GENERAL DISCUSSION OF RESULTS OBTAINED.

Referring again to Table 1, it will be noted that the percentage of infected men in the different mines varies from 16 to 67 per cent. It is not an easy matter to account for this difference, as the factors influencing hookworm infection are many and varied. To simplify explanation only two mines will be compared, the one having the highest and the one having the lowest percentage. Strange as it may appear, there is a striking similarity between these two mines. One would say that these two mines where hookworm infection is most divergent are more alike than any other two mines examined. Salient facts regarding these two mines are assembled in Table 13.

TABLE 13.—Data on the two mines having the maximum and minimum percentages of infection.

	Mine B.	Mine C.
Percentage of underground men infected.....	16	67
Average number employed underground.....	270	260
Are toilet cans underground.....	Yes.	Yes.
Number of months cans had been installed at time of inspection.....	3	6
Number of times per week cans are cleaned.....	2	7
Distance of lowest level along slope, feet.....	3,750	2,850
Number of levels developed.....		16
Number of active working levels.....	13	6
Average temperature, ° F.....	78	75
Average duration of employment of underground men, months.....	26	13
Amount of water removed daily.....	125,000	200,000
Is sewage well taken care of.....	Yes.	No.
Average relative humidity, per cent.....	97	99
Approximate slope of shaft.....	90°	70°
Underground men by nationalities:		
Austrians, per cent.....	9	44
Italians, per cent.....	57	11
Spaniards, per cent.....	12	36
Americans, per cent.....	22	10

The mines practically adjoin each other, and are the two examined, one employing 270 men and the other 260. The sanitary accommodations in both mines are the same and ground toilet cans have been installed in both for about a period. They are both deep mines, and there is little difference in temperature, only 3° F.

Some of the conditions indicate that mine B should have a higher percentage of infection than mine C, because the workings are concentrated, the mine has been worked longer, and has a larger number of total levels and active working levels. More than the average duration of employment of underground men in mine B is 18 months, which is twice that of mine C, and increases the likelihood of infection.

Why, then, are only 16 per cent of the men at mine B infected and 67 per cent at mine C? The following reasons help to explain the discrepancy:

1. Although at present the sanitary accommodations are satisfactory, it is believed that in the past mine B has been kept cleaner than mine C.

2. Mine B makes about 125,000 gallons of water per day, whereas mine C makes about 200,000 gallons. Mine drainage is closely watched in mine B, whereas it is not in mine C.

3. Mine C employs a more transient class of labor, therefore the workmen are more indifferent to the welfare of the mine; many of the floating class apply for work when they are without employment and go into the mine with worn-out shoes and perhaps are careless about bathing and the cleanliness of their shift clothes. The house at mine C had poorer facilities and was not kept so clean as the one at mine B.

4. Mine B has a vertical shaft, whereas at mine C the shaft is at an angle of about 70°. A vertical shaft is probably easier to keep clean and dry, and subjects the men to less possibility of infection when they are raised and lowered than an inclined shaft. In this connection it is interesting to note that mine B, in addition to its principal vertical shaft, has an inclined shaft a few hundred feet away, which penetrates the older workings. In this older part of the mine men work under conditions approximating those in mine C. In mine C 67 per cent, or 44 per cent, were infected; whereas only 39 of the 67 per cent, of the men in the newer part of mine B were infected. This rather confirms the belief that much of the infection is contracted in the shaft.

5. Nationality of labor force is perhaps a factor; of the 270 underground men examined in mine B only 9 per cent were Austrians and 52 per cent were Italians, whereas in mine C 44 per cent were Austrians and 11 per cent were Italians.

commendations as to how hookworm infection can be stamped out in the mines of the Mother Lode are contained in pages 43 to 44. The report will place in the hands of mine superintendents of the United States a knowledge of hookworm infection and how it is spread, so that they will be able of their own initiative to eradicate it if it already exists, or, better still, to install improvements which will prevent its introduction. After all, most of the sanitary recommendations made herein should be followed as a matter of cleanliness and hygiene, irrespective of their relation to hookworm infection.

DIAGNOSIS AND TREATMENT OF HOOKWORM INFECTION

By JAMES G. CUMMING.

As stated in the introduction of this report, it was planned to investigate the prevalence of hookworm infection among the miners of the Mother Lode district, to diagnose fecal specimens from the miners and determine the percentage of infected men.

COLLECTION OF SPECIMENS.

It was decided not to make the microscopical examination of stool specimens in the field, as was the procedure followed by the Rockefeller Foundation, but at the main hygienic laboratory, the bureau of communicable diseases of the California State Department of Health. Consequently it was necessary to devise a convenient container for the collection and transportation of the specimens. The container adopted was a half-ounce wide-mouthed bottle. In the cork stopper of each bottle there was inserted a galvanneal spatula somewhat narrower than the mouth of the bottle. This was used to transfer the specimen of stool. The spatula was greatly to the convenience of collecting specimens. The containers were numbered by first paraffining the exterior, then scoring the number through the paraffin, and finally exposing to hydrofluoric acid. Such a system of marking removes all possibility of error through the accidental removal of a paper label or the obliteration of an ink number. Five cubic centimeters of 5 per cent tricresol solution was placed in each container before shipment to the mine superintendent. This solution disinfects the containers and prevents development of the hookworm eggs.

The containers were shipped to the mine superintendent, who directed their distribution through the mine bosses to the miners. Instructions to transfer a specimen of excreta the size of a walnut to the bottle, then return it to the mine boss. When the containers were returned to the laboratory there was forwarded a list of those submitting specimens with their corresponding specimen numbers, also information about miners for use in filling out hookworm certificates" (see page 41).

DIAGNOSIS OF SPECIMENS.

Although there are various methods of diagnosis of hookworm infection, such as the clinical symptoms, the experimental treatment, microscopic examination of blood, and the microscopic examination of stool, it was decided that the latter method would best serve the purposes of this investigation.

It was found that there was a great variation in different mines in percentage of positive specimens, as well as in the degree of infection. It was therefore advisable, as regards economy, to determine whether direct microscopic examination or centrifugalization in connection with the microscopic examination was preferable.

THE DIRECT MICROSCOPIC EXAMINATION OF FECES.

By the method of direct examination with the microscope an amount of the specimen about the size of a match head is transferred by a toothpick to a microscopic slide, a separate toothpick being used for each specimen. The specimen is thoroughly suspended in six drops of a 2 per cent tricresol and 2 per cent glycerin solution. Such a suspension is uniform and not too thick. The tricresol acts as a deodorant and the glycerin prevents rapid drying, thus obviating the necessity of a cover glass. To thoroughly examine such a slide requires 20 minutes.

THE DIRECT IMMERSION MICROSCOPIC METHOD.

There are two essential differences between the technique of this method and that of the direct method. It was noted by Dr. William Pepper,⁶ in 1908, that the eggs of hookworm, although sticking closely to the slide, do not adhere to any of the many other constituents of stool. This sticky property of the hookworm egg, however, is destroyed in the presence of glycerin, so in preparing a specimen by the immersion technique it is necessary to use only tricresol in the suspending solution. If a suspension of stool is thus prepared in a 2 per cent tricresol solution and is permitted to settle for seven minutes and then the slide is gently immersed in slowly running water, the eggs are found adhering to the slide, while the non-adhesive debris is washed away. If several applications of the suspended stool are applied to the slide and gently washed off, the slide becomes thickly studded with eggs. Finally a 2 per cent glycerin solution may be dropped on the slide and the specimen examined under the microscope.

THE CENTRIFUGAL METHOD.

The procedure by the centrifugal method is as follows: An amount of stool about the size of a small lima bean is transferred to a small

⁶Wiles, C. W., Hookworm disease: U. S. Public Health Service Bull. 32, 1912, 40 pp.

centrifuge tube, which is about half filled with water. tube is closed by holding a square of thin rubber belting opening and the contents are shaken until the material is geneous suspension. Eight such specimens are prepared. first centrifugalization of one set of four specimens, in a centrifuge, the other set of four are centrifuged while the is being prepared for the second centrifugalization. In the time is lost. After each centrifugalization, the supernatant water containing bacteria and light organic matter is poured off. Fresh water is added, the contents shaken and again centrifuged. It was determined by test that the hookworm eggs were completely thrown down on whirling for 25 seconds at 1,800 revolutions per minute. After the third washing the sediment loses its sticky character and is granular in appearance.

MICROSCOPIC EXAMINATION OF CENTRIFUGED SPECIMEN.

The sediment may be directly examined under the microscope. satisfactory results, for it is found that there is little granular material to obscure the eggs. Another method is to immerse the sediment before examination. By this procedure the suspended sediment is permitted to stand on the microscopic slide for several minutes, then poured off, and the slide gently immersed in slowly running water. If the application, sedimentation, and immersion be repeated several times, the chance of making a positive diagnosis in a lightly infected individual is greatly increased.

It will be noted that the centrifugal and immersion methods referred to are in the main those originated by Dr. Williams in 1908.

In preparing the microscopic slide for examination, the cover glass should cover at least the middle two-thirds of the slide, one-third on each side of the center line of the slide. A line of asphaltum made by a glass pencil will keep the suspended specimen within the desired bounds. The specimen is examined under a 10 inch objective. A mechanical stage is essential, so that the entire specimen may be gone over.

For the purposes of the present investigation of hookworm in miners the method of immersion and microscopic examination of the centrifuged specimen has been adopted as the standard procedure. This method is not only the most reliable, but it has the added advantage of being the most economical in the examination of lightly infected specimens.

If, however, 10 specimens from a lot of 100 or more from a certain mine are examined and 50 or more per cent give a positive test by the direct-immersion method, it is more economical to examine the entire lot by this method first and make use of the immersion method for the remainder.

of examining the centrifuged specimen in all those giving a positive test by the direct method.

RESULTS OF EXAMINATION.

The results of the examination of 1,440 specimens are shown in Table 14.

14.—*Results of preliminary examination of feces for hookworm, July 5, 1916.*

Mine from which specimens were obtained.	Total number of specimens.	Number giving positive test.	Number giving negative test.	Percentage of positive tests.
A.....	364	130	234	36
B.....	307	51	256	17
C.....	340	139	201	41
D.....	66	26	40	39
E.....	56	13	43	23
F.....	48	8	40	16.6
G.....	149	46	103	31
H.....	110	31	79	28
Grand total...	1,440	444	996	30.8

ARRANGEMENT AS TO DIVISION OF COSTS AND METHOD AND RESULTS OF TREATMENT.

The mine operators of most of the mines examined have agreed to pay for the treatment of their employees, with the miner the expense of his treatment. That is, the miner pays the druggist for the medicine and the miner pays the physician, but as at some of the mines the physician is a part of the mine organization, the employee receives the entire treatment, including medicine and physician's attendance, without extra cost.

Where there is a division of responsibility for hookworm infection, the miner being partly responsible because of his carelessness and the mine operator because of the insanitary conditions prevailing in most of the mines, it was considered fair that there should be a division of the cost of treatment, thus impressing upon each the fact that his responsibility should be an influential factor in the success of the campaign against hookworm. Furthermore, it was agreed with the mine superintendent at each mine that all employees found to be infected should take the treatment, and that specimens from all new employees should be submitted.

The following formula for hookworm treatment was adapted from the United States Public Health Service:

As a preparatory treatment, at 7 a. m. give 60 c. c. and at 7 p. m. 60 c. c. of a saturated solution of sodium sulphate. The next day give 15 drops of oil of chenopodium at 7 a. m., 15 drops at 9 a. m., 15 drops at 11 a. m. Oil of chenopodium may be given on a lump of sugar or in a capsule. If in a capsule each dose should be accompanied by 15 drops of HCl in a glass of water.

At 1 p. m. give 20 c. c. of a mixture containing 18 c. c. oil and 2 c. c. of chloroform. At 1.30 p. m. give 30 c. c. castor oil and a cup of hot tea.

The above treatment is for adults. The size of dose for chenopodium is as follows: Children 6 to 7 years old, 5 drops; 7 to 9 years, 7 drops; 10 to 11 years, 10 drops; 12 to 15 years, 12 drops; 15 years and over, and under 60 years, 15 drops. The stock of chloroform and castor oil contains 18 c. c. of castor oil and 2 c. c. of chloroform, making 20 c. c. in all, in each dose. For children this mixture is given with plain castor oil as follows: Children 6 to 7 years old, 11 drops or 6 c. c. of the mixture with 14 c. c. of castor oil; 8 to 9 years old, 12 drops or 8 c. c. with 12 c. c. of castor oil; 10 to 11 years old, 15 drops or 12 c. c. with 8 c. c. of castor oil; 12 to 13 years, 20 drops or 4 c. c. with 6 c. c. of castor oil; 14 to 15 years, 25 drops or 16 c. c. with 4 c. c. of castor oil; to adults 30 drops or 20 c. c. of the mixture.

The names of the infected men at each mine were sent to the local physician, and arrangements were made so the physician could give the men, in groups of 10, instructions as to the proper order of treatment. It was suggested to the druggists that the distribution of the medicine could be expedited by inclosing the complete treatment in one package with definite labels indicating the order of taking. The cost of one complete treatment is small and this is paid by the mine operator.

REEXAMINATION OF SPECIMENS FROM TREATED MEN.

When the treatment of all infected men at any mine is completed the physicians send a list of these, and specimens of stool which are again submitted for examination.

The results of the reexamination are reported both to the mine superintendent and the physician. Those miners infected and not treated and specimens submitted for examination.

RESULTS FROM CHENOPODIUM TREATMENT.

Following the first curative treatment, 101 stool specimens have been examined. Of this number of persons 86 (85 per cent) have been cured. Eleven of the remaining 15 were given a second treatment with the result that 5 (45 per cent) were cured. Although the number of cases here reported is not large, yet the value of chenopodium treatment as compared with the thymol is evident. In the final report on the hookworm investigation the California State Board of Health will make a complete report on the results of the chenopodium treatment.

CERTIFICATE ISSUED TO MINERS FREE OF HOOKWORM.

As soon as a specimen submitted by a miner is found free from infection a certificate is issued to him. On the completion of the examination of the specimens from a mine the certificates are issued to all men whose feces are found negative, and as soon as the infected men are found to be cured, as determined by reexamination, they also receive certificates. The immediate issuance of certificates when the specimens are found free from infection arouses interest sufficient to make those not possessing certificates desirous of getting one. The superintendents of all the mines so far examined have agreed to employ as workers in the mine only those men possessing certificates and, further, to insist that all new employees not possessing certificates submit a specimen for examination and, if necessary, be treated.

It is expected that each superintendent will enforce these regulations and that eventually all California miners will possess a certificate. The cards on which the certificates are printed are 4 inches wide by 2½ inches high. Each certificate bears the following inscription:

*Form for hookworm certificate.***CALIFORNIA STATE BOARD OF HEALTH.****BUREAU OF COMMUNICABLE DISEASES.****HOOKWORM CERTIFICATE.**

BERKELEY, CALIFORNIA, .

Date, _____

I am to certify that the excreta of _____ have been examined microscopically and have been found free from hookworm infection.

Director.

Identification.

Age, _____ Height, _____ Weight, _____
Occupation, _____

This certificate is good for one year from date.

(On reverse side.)

HOW TO PREVENT HOOKWORM INFECTION.

Never commit a nuisance in the mine.

Use surface toilets if possible.

Scrub hands thoroughly before eating lunch.

Scrub hands thoroughly and take a hot shower at quitting time.

Boil out shift clothes at least twice a week.

Keep your mine shoes in good repair.

Very truly yours,

JAS. G. CUMMING, M. D.,

Director Bureau of Communicable Diseases.

JOSEPH H. WHITE.

It will be noted that the certificate is good for one year from the date of issue. This period was chosen because it is expected that after the certificates are issued men will become infected before the mine is entirely freed from hookworm contamination. Moreover, the possibility that some of the lightly infected specimens were recognized on the first examination. In view of these considerations it is the intention of the State board of health eventually to require all California miners.

COSTS OF INVESTIGATION.

The costs of the field examination, collection, and inspection of fecal specimens, and of administering hookworm treatment to miners at 8 mines, are as follows:

Costs of hookworm investigations at 8 mines for 444 miners

Expenses incurred by the State board of health:

Traveling expenses.....	
Salary for time of field investigation.....	
500 containers.....	
Laboratory salaries.....	
Literature (including "Special Hookworm Bulletin," report cards, and certificates)	

Total

Examination of specimens:^a

Examining 1,119 negative specimens.....	
Examining 444 positive specimens.....	\$97.

Expenses incurred by the mine operators:

Druggists' bill (444 routine treatments, at approximately 50 cents each).....	222.
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Expenses incurred by the miners:^b

Physicians' fees (444 routine treatments, at approximately 50 cents each)	222.
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Total (exclusive of negative specimens).....

Total expenses (including examination of negative specimens.....)

^a The total 1,563 specimens were examined at a cost of \$0.22 per specimen; the 1,119 negative specimens was \$0.13 per man.

^b Expense per infected miner, \$1.22.

COMMENDATIONS FOR THE PREVENTION OF HOOKWORM INFECTION.

By JAMES G. CUMMING and JOSEPH H. WHITE.

order that the infected mine workers of the Mother Lode be that future infection of the mines be stopped, and that the of contracting infection from the existing infectious areas the mines be minimized, the following 12 recommendations are to the mine operators of the Mother Lode:

Every employee should be examined for hookworm infection. California State Board of Health will supply specimen bottles make the examination free of charge.

Every infected miner should be treated until cured. The California State Board of Health will reexamine all men to determine whether the treatment has been effective.

No applicant for work should be employed until he presents "hookworm certificate" from the California State Board of Health stating that he is free from hookworm infection. Every miner when free from hookworm will be issued a certificate. The California State Board of Health will examine specimens of feces from applicants free of charge. The applicant may be employed tentatively while the result of his examination is pending.

All mine workmen should be reexamined about one year from date of the issuance of the "hookworm certificate." It is advisable for the bosses, repair men, and skip tenders to be reexamined months after date of issuance. The California State Board of Health will make these second examinations free of charge.

The toilet accommodations in the mines should be improved by using toilet cans similar to those shown in figures 1 and 2 (pp. 13 and 14), and these cans should be emptied and cleaned at least three times a week.

Sanitary water-flush closets should be installed at the surface convenient to the shaft collar and be maintained in a clean, sanitary condition, and drastic measures should be adopted to prevent defiling these conveniences.

At underground stations where the workmen eat dinner facilities for washing the hands should be installed, benches allowing a seat for each man provided, garbage cans for food remnants

installed and their use enforced, and the stations well lighted and maintained in a clean, sanitary condition.

8. Rats should be exterminated from the mines. It is estimated that if the mines are kept as clean as possible the rats will starve out.

9. Sanitary drinking fountains or sanitary water kegs should be used in the mines. If kegs are used, they should be sterilized with live steam each night without fail.

10. The drainage system in the mines should be carefully maintained, the mine water being prevented as far as possible from flowing from one level to the next. The drainage ditches on the levels should be kept open, thus keeping the levels, and particularly the foot walls, dry and in good repair and free from muck.

11. Certain improvements should be made on the water supply. The houses should be scrubbed out with water and a disinfectant after each shift. Additional showers should be installed where necessary, all wash basins discarded, the washing trough made permanent, and the hot and cold water connected to a common mixer, thereby allowing the miner to wash in flowing water of any desired temperature, and facilities provided so that the men may keep their "shift" (working) clothes.

12. The miners should be instructed in the causes, nature, and prevention of hookworm, so that they may know how hookworm spreads and intelligently subscribe to the remedial measures introduced. Some educational work has already been accomplished by the instructions on the back of the "hookworm certificate," the formal talks given by the field investigators, and by a special circular on hookworm, written by the authors of this report and approved by the California State Board of Health. Arrangements are now under way for holding miners' rallies in important mining districts where illustrated talks will be given on hookworm infection, prevention and cure.

* Cumming, J. G., and White, J. H., Prevention of hookworm: Special Circular, California State Board of Health, May, 1916, p. 564.

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Bulletin 140

DEPARTMENT OF THE INTERIOR

FRANKLIN K. LANE, SECRETARY

BUREAU OF MINES

VAN. H. MANNING, DIRECTOR

**OCCUPATIONAL HAZARDS AT BLAST-FURNACE PLANTS
AND ACCIDENT PREVENTION**

**BASED ON RECORDS OF ACCIDENTS AT BLAST
FURNACES IN PENNSYLVANIA IN 1915**

BY

FREDERICK H. WILLCOX

**This report was prepared under a cooperative agreement with the
Pennsylvania Department of Labor and Industry]**



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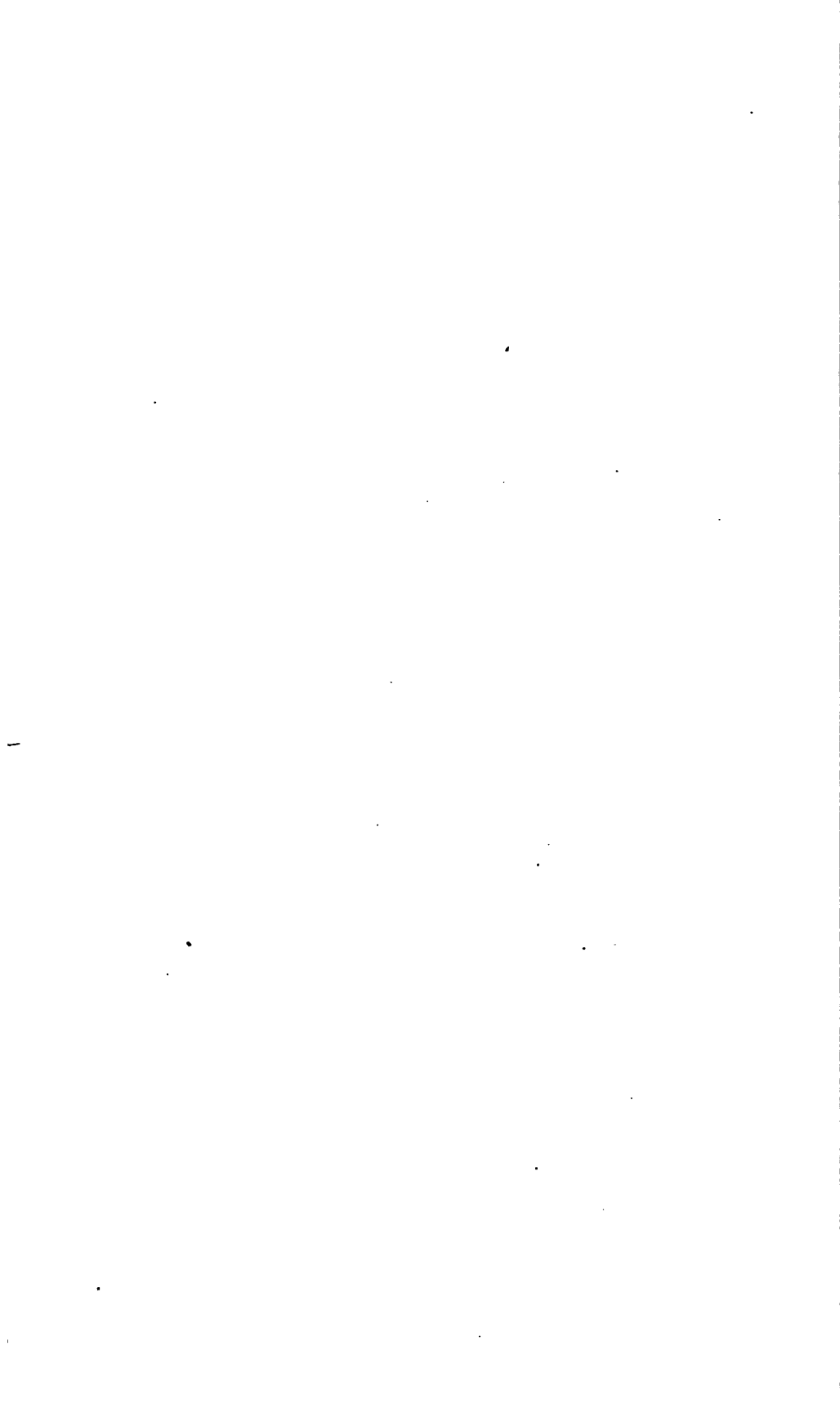
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PREFACE.

The information presented in this report is the outcome of a detailed investigation made at the blast-furnace plants of Pennsylvania during the year 1915 by the inspectors of the department of labor and industry of that State and by F. H. Willcox, metallurgical engineer of the Bureau of Mines. Although accident-prevention methods have been in effect at the majority of plants in the State for a considerable number of years, and although the State department for several years has had its inspectors at the plants to examine plant conditions and to investigate serious accidents, the hazard of blast-furnace work, as indicated by the insurance rates for the workmen, remains among the highest of any of the industries of the State.

It is hoped that this detailed review of the causes, circumstances, and prevention of accidents will be of assistance to the executive officials of the plants and to the employees by making clear the responsibility for accidents, by emphasizing the probability of repetition of many accidents unless care is taken, and by demonstrating the practicability of safeguards, thus resulting in a lessened number of injuries and fatalities.

Though effort has been made to reduce the report to a minimum, it still is large, and there is considerable repetition of discussion of many types of accident as they occur at different places in the plant. However, such repetition has been retained in the belief that it will serve to emphasize the importance of attention to accidents that occur repeatedly and contribute the greater share of the charges for disability and compensation.

VAN. H. MANNING,
Director.



OCCUPATIONAL HAZARDS AT BLAST-FURNACE PLANTS AND ACCIDENT PREVENTION.

By FREDERICK H. WILLCOX.

INTRODUCTION.

In the past the blast-furnace industry was under the stigma of being one of the most prolific sources of killed or seriously injured and permanently disabled workmen of any of the industries of the country. In the popular mind, labor at blast furnaces was little less hazardous than mining, powder making, or railroad work. This impression was based not so much on definite knowledge of the number actually injured and the causes of injury as on the spectacular nature of the accidents that from time to time were described in the press. As with a mine disaster or a railroad wreck so with many furnace accidents, the number fatally injured in such accidents constitutes only a minor proportion of the workmen who suffer more or less serious injuries. However, the reader of the daily paper receives a distinct impression of frightful and sudden disaster which is apt to be associated henceforth in his mind with work at blast furnaces. It is true that in the past this impression has not been without warrant, for from the time of the advent of Mesabi ores, taller furnaces, and faster smelting, which began in the early nineties, the newspapers have chronicled a large number of blast-furnace disasters.

Blast-furnace men have never assumed to tolerate failures in construction or defective control of furnaces. But the admitted hazards of the work are many, and the slow development of improvements in the construction and control of the furnace and its auxiliaries have occasionally given rise to a tendency to accept as a part of the day's work not only the unforeseen and difficultly controlled accidents but accidents whose recurrence might be stopped by the adoption of suitable preventive measures.

In the smelting of iron, even more than in many other industries, safety is inseparably related to efficiency in production, and the accidents strictly incident to work about the furnace are proportional in some degree to the success with which obstacles to smooth working are overcome. This relation does not hold, however, in regard to such accidents as are incident to hand labor, falls, falling and flying objects, machinery, and the use of hand tools.

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The author desires to express his obligation to the officials of furnace plants in the State of Pennsylvania for their hearty cooperation while the field work of the investigation was in progress and to the many furnace men and safety inspectors at the plants for their free and helpful discussions of plant difficulties and hazards and the methods of meeting them.

DEVELOPMENT OF THE MODERN BLAST-FURNACE PLANT.

For a considerable number of years the all-important aim of furnace operators was tonnage. The reasons for centering attention on large output need not be discussed. Immense ore supplies, enormous prospective demand for iron, the cost of labor, and the economies to be effected by working on a large scale were probably the controlling motives. In justification of their views, blast-furnace operators can point to the high degree of efficiency developed at furnace plants as regards application of power, the utilization of the machinery, and the high output per man employed. This development, however, was not well synchronized; it required faster work with novel and dangerous equipment, the handling of increasingly larger quantities of bulky material and of product, the handling of the latter being peculiarly difficult, and was accompanied by a change in the character of the labor supply. Difficulties in furnace operation were increased by the necessity of using larger and larger quantities of increasingly leaner Mesabi ores.

RESULTS OF SEEKING HIGH TONNAGE.

As a result of the demand for tonnage and lower smelting costs, furnace shells were built increasingly higher and of larger capacity and linings were made thicker to give longer service, but some details of furnace design received inadequate attention; the general construction of the furnace was not always such as to conduce to security. A conservative statement would be that the mechanical engineer had usurped the place of the furnace engineer and had developed the furnace with little appreciation of the duty that was to be required of it.

The use of Mesabi ores had shown that mixtures containing them would produce frequent slips violent enough to blow off the furnace top. Instead of providing tops of sufficient strength to withstand the pressures generated by a slip, reliance was placed in explosion doors to relieve the pressure developed by the slip. Inasmuch as increasing the height of stack had led to larger tonnage with an attendant decreased coke consumption, the natural trend of design was to build higher and higher shells until an extreme of 110 feet was reached. The lining of the shell was increased in thickness to a maximum of 6 feet in order that the furnace might be driven hard for maximum tonnage without frequent and annoying delays for relining.

A very large furnace, driven with a tremendous volume of blast, having been evolved, it was found that design and capacity had for the time being gone beyond the limit of control of the furnace operators. The increased height of the furnaces and the finer ores used led to slips of undreamed-of violence, to showers of red-hot burden from the explosion doors, and to blown-off tops. The channeling effects and the pressure of the blast and the irregular working consequent on slipping rapidly wore out the lining. Moreover, there was a determination to work the lining to destruction and a disinclination to go out of blast until the point of absolute failure had been reached, because of the cost of relining and the time required for it. Consequently linings were allowed to become badly eroded, so that scaffolding and severe bosh slips resulted. As the brickwork of some boshes was insufficiently tied in and was weakly banded, occasional bosh failures became features of practice, and as the design, cooling capacity, strength, and brickwork of the hearth in many furnaces were inadequate and faulty, breakouts were not infrequent.

These conditions are now a matter of history, but their discussion is necessary to an appreciation of the contrast between the furnace of 1895 and that of 1915. Far from being an indictment of the men who engineered the development, this discussion is rather a statement of the handicaps they overcame. The past 20 years has covered a period necessary to development. Furnace men to-day reviewing that development have not infrequently stated that during those days a strict application of "safety first" to design and construction would not have resulted in as rapid advancement nor in the present degree of security.

Progress and courage in design, based on the practice since 1895, have to-day resulted in furnaces with closed tops strong enough to withstand the pressure from any slip. There are still many furnaces in which there is danger from slips, but the danger is being much lessened by more regular driving and better adaptation of design to

the ores, coke, and blast temperature used. Linings are made much thinner and are not used to destruction, although some thick-lined furnaces remain in blast, and there is a disposition to use them in spite of the danger of failure, because of the time and expense required for relining. Bosh breakouts have become extremely rare—practically unheard of—thanks to additional strength and reinforcement at each relining and in each new furnace. Hearth breakouts also have become rare, owing to better construction and more ample cooling, but occur at infrequent intervals, and seemingly will persist because of the peculiarly combined effects of blast erosion, the chemical attack of slag and iron, and the pressure of the liquid iron on the brickwork. In conceding and emphasizing the splendid progress in design and strength which in modern blast-furnace plants obviates most of the causes of the accidents that have given the industry a bad name, one should remember that the influence of extreme tonnage demands remains and that furnaces of antiquated design, though doing efficient work, are contributing a certain proportion of accidents that are absent in newer and stronger types.

ADVANCES IN BLAST-FURNACE PRACTICE.

As the disadvantages of hard driving became evident to the men in charge of plants that were using higher proportions of Mesabi ores each year, and as the greater regularity and the economy in fuel and in lining from easier driving became apparent, there was a swing at many plants to extremely moderate driving—"slow driving." This tendency was accelerated by the let-up in production in 1907, by the great increase in productive capacity of the country, and by the larger output of basic iron, it being more difficult to produce good basic iron than good iron of higher silicon content in a furnace working irregularly, because of slips. Contemporaneously, there came the possibility of using a hotter blast, because of the smaller volume of blast with slow driving and the use of cleaned gas for the stoves.

The result of this change in practice in many plants was renewed heavy slipping, scaffolding, and other irregularities of operation, because the lines of practically all stacks were unsuited to the use of higher heat, or because this higher heat was unsuited to the type of slag that had been made, or because lines, slag, coke, and heat were all unsuited to each other. Also, the volume of the blast was so reduced that it failed to penetrate the burden thoroughly, but worked along the walls, causing erosion of the lining and insufficient reduction of iron. About the same time other difficulties were introduced by attempts to use by-product coke. In short, the attempts at new practice gave rise to almost as much irregularity in operation as

former practice had caused, so that furnace accidents persisted to an extent that was lessened only by improvement in construction.

Within the period 1910-1915 much definitely satisfactory progress was made in obtaining the type of practice sought since 1907-8, namely, the use of high heat and moderate driving. The results may be described as moderately fast driving, the use of moderately high heat, radical changes in stack lines, and the production of a less limy slag. A corollary of this advancement has been a marked decrease of the accidents peculiarly due to slips and other irregularities of operation. Probably quite as much significance attaches to improvement in practice as to betterment in the mechanical design of the furnace.

APPLICATION OF POWER AND HANDLING MACHINERY.

With the increase in the capacity of furnaces there was need of more adequate methods of handling ore, coke, and limestone, and of disposing of iron and slag. A plant operating four furnaces requires each year about 1,000,000 tons of ore all of which has to be unloaded inside of 120 shipping days, so that, if old methods were used, an average of 170 cars would have to be unloaded by hand every day. Such a task was not easy when the number of loaded cars averaged the same each day, and was rendered extremely difficult by the occasional slackening of shipments for a couple of days, the resulting rush, and the congestion of cars. The labor supply was frequently wholly inadequate for taking care of such situations, particularly as some of the men were never sure of employment from day to day, to say nothing of employment in winter, and were constantly shifting to other work.

The development of the mechanical car dumper was necessary to keep the furnace running, and characteristically this appeared at an opportune time to compensate for a diminishing and unsatisfactory labor supply. The earlier development of the ore bridge or crane was due to somewhat similar causes—the impossibility of handling and stocking a high pile of ore in a restricted space and the difficulties in getting the ore from the stock pile to the bottom of the furnace by barrows or by steam cranes. Bins for the ore were introduced about 60 years ago. The larry car, operating on top of the bins, was a logical sequence to the bridge, for it facilitated the transfer of ore from the dumper to the pile or from the pile to the bins without consuming the time of the ore bridge in runs up and down the trestle to get from the stock pile ore of the grade desired in the bin, which frequently is some distance from the pile. In the stock house, a similar car, equipped with scales, was a natural development, as it replaced 12 or more men pushing barrows from the bins to the foot of an elevator. To enable a small crew to get stock from the

bins, mechanically operated bin doors and feeders were installed. The substitution of skip inclines, mechanically operated furnace bells, and automatic dumping was the last step in reducing hand labor in moving the ore, and permitted the number of laborers to be reduced more than 75 per cent. But with fewer minor injuries because of this cutting down of the laboring force came the hazards incident to using somewhat intricate handling machinery, designed primarily for power and capacity and with minor regard to safety, operated at high speed, and hurriedly repaired in case of a breakdown. Moreover, this machinery used electric current obtained from trolley wires or third rails, in some places inadequately guarded and often placed with too little regard for inaccessibility from accidental or careless contact.

As the coke and the limestone, as well as some of the iron-bearing by-products were unloaded on the trestle by hand, some of the men were subjected to the risk of being hit by material dropping from the ore bridge, or of being struck by the larry car running up and down along the bins, as well as to the added hazards of operating and oiling the machinery and those peculiar to hurry-up repair jobs that were frequent because the machinery was not perfectly developed.

Progress in the blowing and power equipment and water-supply pumps had to keep pace with tonnage demands, but was attended with fewer accidents than was the development of other equipment. One reason was that similar machinery was being developed in other industries and was better standardized than ore-handling equipment; another reason was that the workmen operating such machinery were more skillful and intelligent. A new appliance, the gas engine, added to the complication of handling the gas.

The cast house with its beds of chills or molds proved expensive and inadequate for handling large outputs. The invention of the pig-casting machine and, at steel plants, the carrying of the molten metal in track ladles to the mixer eliminated the casting beds. Trap ladles were also introduced to remove the molten cinder instead of running it on cooling beds and then forking it into cars, or the cinder was run into a granulating pit and then removed with a crane.

The place of least change has been immediately about the blast furnace where the hand tools of the seventies and the same hard and trying work are still evident, but invention and better practice have obviated many of the severe tasks and dangerous risks. Steam, air, or electric drills, electric and oxygen burners, cast-iron skimmers, and clay guns are among the outstanding improvements about the furnace. Improvements in the stoves have little effect in complicating operation, as the improvement has been in size and in lining, but the gas-main system with its growing intricacy of washing equipment and of connections has demanded adequate control and skilled super-

vision, especially on stops, blowing in, or blowing out. A patented method of applying steam to gas systems has, however, lessened the hazard of explosions in such systems.

CHANGING CHARACTER OF LABOR.

The development of mechanical appliances and the essentially dirty, hot, and rough nature of much of the work have caused a change in the character of the laborers at blast-furnace plants. For 30 years, in all but a few localities, American-born workmen have steadily decreased. The predominating nationality has changed from time to time, but laborers from the most recent immigration have always been the predominating element at blast furnaces. The increasing use of machinery requires more skilled workmen, but ordinary laborers must still do hard, rough work, such as handling material, shoveling, and pushing wheelbarrows.

Labor of the next higher order is needed for such work as cleaning stoves, cleaning iron ladles, dumping cinder ladles, working about the pig machine, and cleaning and making up iron runners and cinder runners. By "making up" runners is meant claying, grouting, or sanding the cleaned runner preparatory to the next cast, adjusting the gates, and warming cold or wet spots. From these jobs men are promoted to more responsible duties. They become stove tenders, keepers, or water tenders; are put in charge of the simpler machinery, such as pig machines or charging cars; oil and wipe in the engine room; or work in the millwright, rigging, or pipe-fitting gangs.

All of these jobs involve laborious, fast, hot, or hazardous work; much of it, in the opinion of many operators, demands men of stolid and phlegmatic nature who can be easily controlled by shift foremen, and accept hard or trying work without protest.

In the past a man who was temporarily disabled by an accident usually had to accept his loss of time and income with no recompense from his employer or the State. Although this condition existed in practically all industries, it bore more heavily on furnace men than on those in many other occupations, because of the greater hazards at blast-furnace plants.

Other factors tending to discourage more intelligent men from taking up blast-furnace work have been the prevalence of the seven-day week, which is more characteristic of blast-furnace plants than of other iron and steel operations; the low wages, the average wage being lower than in any other branch of the industry; and the fluctuating production. The last involved the certainty that at recurrent periods the bulk of the furnace force, except the most skilled, would be out of work or would have to accept jobs with diminished pay.

It is a matter of record that the working conditions mentioned have been unattractive to American-born workmen, and as conditions improved and as immigrants from the British Isles and Germany decreased, the operators were forced to employ other men. As the number of Poles, Slavs, Italians, Magyars, and men from the Balkan States entering the labor gang and working up into the furnace and other crews increased, both the older hands and recent immigrants from northern Europe declined to seek employment among them. This racial antagonism worked in other ways; strikes by employees sometimes caused a complete change in the character of the force, English-speaking stockhouse men, riggers, and crane runners being replaced by men from southern Europe.

One result of such changes in the character of the labor has been to make accident prevention more difficult. Only 5 to 10 per cent of the immigrants from southern and southeastern Europe have come from manufacturing industries, as against 40 per cent of the earlier immigrants. About 70 per cent of the recent immigrants were farmers. Their speech includes many different languages or dialects of the same language, and they may be so divided by racial antipathy as to lose common interest and fellow feeling. Frequently 50 per cent can not speak English, though they commonly understand a few terms used in directing the work.

SIGNIFICANCE OF PRESENT TYPE OF BLAST-FURNACE LABORERS.

Thus instruction by signs and books of rules becomes most difficult, and it is always questionable whether workmen understand each other or the instructions of their foremen. It may happen that they fail to heed a cry of warning when quick action is needed to escape some sudden danger, particularly when the danger is obscure or not self-evident. This language handicap of the new men, their unfamiliarity with the dangers, their ignorance of the equipment with which they must work, coupled possibly with less alertness, render them peculiarly liable to injury.

It is only to be expected, therefore, that a large number of accidents will result from employing such men in work involving the moving of large quantities of material, the handling of great volumes of poisonous and explosive gases, the use of blast at high temperatures and pressures, the pouring of fluid iron and cinder, and the operation of novel machinery.

GENERAL CHARACTER OF BLAST-FURNACE ACCIDENTS.

As is pointed out on later pages, most of the accidents discussed in this report did not result from some exceptional disaster, such as a slip, breakout, or explosion, but arose from a variety of causes

which, except for hot-metal and cinder burns, are found in operations not essentially typical of blast-furnace work, although of necessity connected with it. The difficulty introduced by the employment of men from southern Europe is not essentially different from that in other industries in which rough unskilled labor is usually done by such men.

Thus the subject of accident prevention about blast-furnace plants is not as unique as many persons have believed it to be. Because of the complexity of the smelting process there is great possibility of danger to many men should one man ignorantly blunder or be neglectful in performing his duties. Consequently the rarity of disastrous accidents attributable to the fault of the furnace force speaks much for the skill, intelligence, and resourcefulness of the men who comprise that force, drawn, as they are, from material seemingly so unpromising. It suggests that the members of that same force can acquaint themselves with and protect themselves against the hazards of their work. In any occupation a man assumes more or less risk, and in working at a blast furnace he assumes a risk that is rated by insurance companies as among the highest. Whether the risk at the place where a man is employed about the furnace is small or great, he should know of it and be able to take special care to avoid it.

Even though he is compensated for injuries lasting more than 14 days, the burden of financial loss falls chiefly on the workman, for the majority of accidents cause a loss of time less than the minimum compensation period, and consequently a complete loss of wages. No blast furnace can run without making some offgrade iron; similarly, accidents must be expected. However, there is no more necessity for an accident rate of 250 per 1,000 men than there is for a production of 25 per cent of offgrade iron, and the inherent risks are great enough without the workmen being ignorant of the nature and causes of the common accidents.

Practically every accident about a furnace is a repetition of a similar accident at the same plant or at other plants. A large proportion of these accidents can be avoided by the exercise of ordinary skill and prudence, provided the men are thoroughly acquainted with the causes of the accidents and the hazards of their occupation. A smaller proportion of blast-furnace accidents occur through the fault of the employer, being due to lack of safeguards, deficient equipment, and failure to give proper instructions, but even in these accidents many injuries may be avoided by pointing out the hazards and insisting on carefulness. Many accidents are ascribable to the trade risk, but a knowledge of their character enables workmen to anticipate in some degree the possibility of such accidents and to exercise a greater control over their occurrence.

REVIEW OF BLAST-FURNACE ACCIDENTS IN PENNSYLVANIA IN 1915.

To afford information, accidents during 1915 at blast-furnace plants in Pennsylvania are comprehensively discussed in the section following; mention is also made of characteristic but rarer accidents not brought out by this review. The discussion should show each man, as well as the employer, the causes responsible for accidents in his occupation. On the theory that what has happened shows what may happen or will happen, the value of this report should lie largely in the remedial action suggested. To increase the value of the report the preventive measures used by various companies in the State are mentioned.

In the review following the accidents have been classified according to the place in the plant where they occurred. They have been further subdivided according to causes.

In the descriptions of accidents the seriousness of the injuries is designated by letters, as follows:

- A. Injury resulting in death.
- B. Injury causing disability of more than 30 days.
- C. Injury causing disability of more than 14 days and less than 31 days.
- D. Injury causing disability of more than 2 days and less than 15 days.

CAR DUMPER AND TRANSFER CAR.

The hazards at this equipment consist chiefly of injury from the mechanism of the dumper, from railroad equipment, and of those injuries that are encountered in hand labor and where hand tools are used.

CAR-DUMPER MECHANISM.

The injuries caused by car-dumper mechanism are mostly encountered in stepping on or off the car-dumper cradle while it is in motion. Examples follow:

1. Card boy, after removing card from car, jumped from the cradle after the operator had started holsting and was thrown from framework of dumper. D.
2. Brakeman stepped on cradle while it was still going down, to climb on empty car. He slipped, fell, and was caught beneath the down-moving cradle. C.

RAILROAD EQUIPMENT.

Railroad equipment at car dumpers causes serious injuries, as is shown by the following examples.

1. Car rider was dropping car down incline from dumper; brake did not work, and man was thrown beneath car when it struck the empties. A.
2. Temporary car-dumper man was standing at cradle to adjust coupler on empty, when a new car was pushed up on cradle, knocking man down and running over his body. A.
3. Laborer was cleaning track at car dumper when transfer car struck him, amputating arm. A.
4. Laborer was in car cleaning ore out when empty from dumper struck car, causing man to fall through door. D.
5. Brakeman jumped from moving car he was dropping from dumper and fell beneath trucks. A.

INJURIES IN HAND LABOR.

The majority of injuries are to be found in hand labor and allied occupations. The seriousness of the injuries is, however, relatively slight, as compared with those received from railroad and car-dumper equipment, as the following examples show.

1. Laborer was standing on plank over larry bin and barring sticky ore from bin into larry car. Slide of ore caught bar, threw him off balance, and he fell into chute. B.
2. Laborer turned ankle in stepping on rail. D.
3. End door of car fell on laborer's foot. D.
4. While men were throwing ore cleanings from one car to another, man passed between cars and was struck on foot by lump of ore. C.
5. Infection of eye due to neglect to have dust removed. D.
6. Laborer allowed tie to drop from his hands onto his foot. B.
7. Laborer jammed fingers between jack and third rail. D.
8. Laborer struck foot with pick. D.
9. Laborer was cleaning ore from car while it was held inverted on cradle; lump fell from car onto his knee. C.
10. Lineman was repairing electric switch; screw driver slipped and caused electric flash by short-circuiting switch poles. D.

Regarding accident 10, work on a switch should not be undertaken until the fuses are removed from the lines and the switch is dead. In addition the screw driver should be kept wound with adhesive tape down to the bit, in order to prevent short circuits should the tool slip when used on uninsulated switches.

A little less than half the men injured about the car dumper had been employed there less than six months, and ignorance of the hazard of the work and lack of skill may have been responsible for the injuries to these members of the force. As will appear repeatedly, however, and as indicated here, the relative number of experienced and inexperienced men injured does not show that with hand labor and hand tools familiarity with the work offers any pronounced

likelihood of fewer injuries. As will become evident, personal caution and skill, which will be acquired under supervision in six months almost as well as in six years without supervision for their development, are requisite for preventing these accidents.

For men employed about the dumper and as car riders, the advantage of experience is more marked. Barring ore in bins requires some experience or special instruction to avoid getting hurt when the bar is caught in a slide of ore. More care is needed in selecting men to drop cars from car dumpers and in work about dumper cradles. If men accustomed to railroading can not be employed, then those selected for the work should be able to size up a dangerous situation quickly and to take the steps necessary to avoid injury, and also should be able to acquire the skill necessary for work about moving cars.

Getting on and off car-dumper cradles while they are in motion is usually forbidden, but is often practiced by employees and ignored by foremen, especially when work is rushed to avoid demurrage charges on cars. The accidents described show the danger of the practice, and also the danger in making a rule and then allowing it to become a dead letter, thereby bringing other rules into disrepute or at least tempting employees to disregard them at their discretion. To warn men whose duties take them on the dumper cradle out of sight of the dumper operator, some plants place a gong in a convenient place, by which the operator may be signaled when the man is clear of danger. The men concerned are commonly the car boy, the brakeman who "sprags" the car, or the brakeman who adjusts the couplings. The cradle is not operated nor are cars brought on the dumper until the operator knows that the men are out of danger. Work about railroad equipment is admittedly hazardous and when coupled with the operation of a car dumper may become more so than usual.

In cleaning cars after they are dumped there are two practices: (1) The cars are cleaned while in the dumper and inverted over the larry-car bin. To accomplish this most safely, a substantial platform with railings and high toeboard toward the dumper is recommended. A variation of this method is to clean the cars from the cradle itself, while it is tipped over the ore yard, but this method subjects the men to the danger of falling lumps of ore and in "rush" work, the operator is apt to start the cradle back before the men are off, the car cleaners stepping from the cradle to the cab platform while the cradle is tilting back. (2) A better plan, if the dumper operates along stock-yard walls, is to clean the cars after they leave the dumper. If the yard is large enough for spur tracks, the cleaning should be done on the spur after it is filled, no empties being allowed to bump into the cars being cleaned. If only one track leads from the dumper, the car

cleaners should never work in the first six cars next to the dumper, and the farther back they work the better, as they are warned by the noise of the first bump to hold tight before the shock reaches the car they are in. The brake of the last one or two cars of the string should be set up to retard the kick. If men are available, it is better to shovel the ore over the side of the car, rather than to open the drop doors, the shovelers thus being safe from falling through the doors and possibly being run over. Ample side clearance should be provided for car riders to jump from moving cars, and enough men should be provided so as not to require one man to adjust the coupler and to "sprag" the cars.

Ore transfer cars should be provided with an effective warning signal. Electric horns have been used, but they do not appear to be as effective as anticipated, because their sound is obscured by mill noises more than is a whistle. A 2-inch air whistle with a distinctive sound is probably the most effective warning. Gongs are not especially suitable for transfer-car service. Fenders that will lift a man from the rails and push him aside should also be provided at both ends of a larry or transfer car. There is pronounced need for the design of improved fenders for larry-car equipment, as practically all in use are more likely to mangle the man struck than to ward him off.

A car dumper with a guarded platform at the hopper and at the sheave wheels is shown in Plate I, A.

ORE YARD AND ORE-BRIDGE CRANE.

The hazard encountered at the ore yard and the ore bridge lies largely in the operation, maintenance, and repair of the electrical and mechanical equipment.

1. Employee was descending ladder on leg of ore bridge when adjacent bridge bumped into it, catching man between two cranes. B.

2. Electrician making repairs to hoist motor kept hand on shaft when signaling operator to hoist, and hand was drawn into mechanism. B.

3. Ore-bridge operator was turning down grease cup on hoist while it was in operation and sheave wheel caught glove, drawing arm into mechanism. B.

4. Millwright was making hitch when craneman raised a little on hoist, catching finger. B.

5. Laborer attempted to dislodge a lump of ore from the jaws of a bucket, when the jaws closed, crushing hand. B.

6. Transfer-car man was attempting to straighten up ball attached to ore bucket when the ball fell onto legs. B.

7. Hooker-on, attaching hook to ore bucket, kept hand on ball as bucket was raised, and caught hand between ball and bail of adjacent bucket. D.

8. Ore-bridge oller pulled electric switch and received electric flash in eyes. D.

9. Electric repair man in putting in fuse caused short circuit, and flash burned eyes. D.

10. Craneman threw switch, causing fuse to blow out, burning hand. D.
11. Craneman connecting wire for electric light caused short circuit, burning hand. C.
12. Rigger was taking strut from bridge with crane. The strut swung and knocked man's leg against rail girder, amputating leg. B.
13. Laborer was in car in which grab bucket was scooping ore. Operator did not see man and hit him on head in lowering the bucket. D.
14. Trestle man was sitting on side of car directing unloading of car with grab bucket, when bucket swung against leg. C.

When work is being done on cranes or runways where other cranes are running, the idle crane should be protected by track torpedoes placed 25 to 75 feet away, depending upon the effectiveness of the brakes. An ore bridge equipped with safeguards is shown in Plate I, B.

The electrical accidents emphasize the need of safety-switch boxes which will prevent flashes from burning the operator when the switch is operated. Fuses should be similarly inclosed or placed so far away from the switch that the flame or flash from the fuse blowing can not injure the operator. For replacing fuse plugs, fiber or other insulated tongs should be provided. It is probably well to insist that the work of installing lights or making connections to electric circuits be done by electricians, as the work is apt to prove hazardous to inexperienced men, although not difficult or dangerous in the least for skilled workmen.

The other accidents disclose a variety of unsafe practices. It is almost a certainty that men who persist in oiling machinery in motion will eventually suffer accident. As regards ore-bridge cranes, at least, there is no necessity for such practice. When practicable to do so, the switch controlling the motor should be thrown open and locked whenever the crane hoist or trolley does not require occasional movements to bring parts under repair within reach. When such movement is necessary, men capable of the duties of electrician or millwright obviously should know enough to get into a safe place before giving any signal for operation that will place themselves in jeopardy. Crane operators should not hoist until the hooker-on has given a signal to do so, and, just as important, the hooker-on should not keep his hand between a chain, cable, or hook and the object to be hoisted after giving the hoisting signal, nor between the material to be hoisted and any object between which he may be caught when the load swings. The signal to hoist should always be given by the hooker-on. These are elementary precautions, and self-evident, but they are repeatedly and persistently ignored in the operation of cranes.

The last two accidents serve to indicate the need of enforcement of practices that will prevent men from remaining inside cars in which ore grab buckets are being used. When it is necessary for men to



A. CAR DUMPER WITH GUARDED PLATFORM AT HOPPER AND AT SHEAVE WHEELS.



B. ORE BRIDGE EQUIPPED WITH SAFEGUARDS.

A, Guard for wheels; B, device for clamping bridge to rails; C, guard for driving gears.

get inside to shovel up the ore for scooping, the crane operator should not lower the bucket until the men are out of the car, or at least in the opposite end of the car from where the bucket is to be lowered. In using hoist tackle in removing and replacing heavy objects there is the same danger of a side swing, and strong guy lines should be used to retard the swing and to control the path of the hoist, and the men in whose hands the guy line is placed should take a wrap or turn of the line about some convenient strong object.

The other accidents are illustrative of the lack of thought of either the foremen in not warning men of unsafe practices or of men in disregarding the self-evident dangers of work with this equipment; they show the need of teaching caution even to men who may have come through some years of experience without serious injury, some of the injured having had as much as 10 years' experience.

FALLING AND FLYING MATERIAL, FALLS OF WORKERS, AND MISCELLANEOUS ACCIDENTS.

1. Laborer was unloading ore from stock pile into buggy when ore rolled down pile onto leg. D.
2. Laborer was breaking lump of ore when piece flew and struck hand. D.
3. Laborer got in way of stone breaker, and was hit on hand with sledge. D.
4. Craneman was cutting hand leather from scrap belting and knife slipped and cut him. D.
5. Repair man missed his grip on rung of ladder while climbing up leg of ore bridge and fell about 20 feet to track rail. B.
6. Oiler of ore bridge, in coming down ice-coated ladder of ore bridge, slipped and fell about 18 feet onto girder of ore bin. B.

Accidents like the last two, which resulted because of a lack of a safety cage, must be attributed to the negligence of the company, as safety cages have been for some years standard equipment on all ladders more than 10 feet high. On the recently erected ore bridges steps rather than ladders are used.

Slivers and fragments flying from ore or limestone being broken with sledges have more than once caused loss of eyesight, and as flying material is to be expected in such work, the men assigned this work should be given goggles and compelled to wear them. There is no such handicap in their use as at times occurs in cast houses and at pig machines because of the presence of steam.

A hazard that exists where men haul ore from the stockyard to the furnace is the danger of undercutting a pile of frozen ore and having the pile collapse and bury the workmen. Foremen should exercise close supervision of the work where charging is still done in this way.

The accident caused by the knife slipping while a man was cutting a hand leather is trivial, but could easily have been serious. When

safety hand leathers are furnished, as is done by a large number of companies, they eliminate a variety of hazards incident to the preparation and use of hand leathers from scrap belting or leather.

TRESTLES.

Flying material, falls, hand labor and hand tools, and railroad equipment comprises the outstanding causes of injuries on ore trestles.

FALLS.

1. Carpenter tripped on loose board in walk and fell to yard. D.
2. Stock unloader, in climbing out of car, slipped and fell. D.
3. Ladle chaser fell from trestle walk to yard, 18 feet, fracturing arm and leg. B.
4. Laborer standing on ore in car was carried into bottom of car when ore collapsed. D.
5. Car unloader was standing on plank along trestle unloading car when plank overturned, causing fall. B.
6. Bin feeder was barring ore in bin when slide of ore caught bar, throwing man into bin. D.
7. Laborer stumbled over coke fork and fell into coke bin. C.
8. Bin man stumbled over loose plank in trestle walk and fell into bin. D.
9. Stock unloader was jumping from rail to rail over trestle bin and slipped, cutting leg on rail. D.
10. Rigger was standing on ore in bin when ore was drawn in the stock house from bin; collapse of footing caused him to be carried down in bin. D.
11. Laborer was standing on ore in car, barring ore through doors, when ore fell in, carrying man into ore bin, causing internal injuries. C.

Although none of the accidents in this classification resulted fatally, there have been in previous years several instances in which similar accidents resulted fatally. Men should not be allowed to get on top of carloads of material before the drop doors are opened. When men must enter a car to unload it, they should be warned to keep away from the edge of the ore toward the drop door, as the ore may slip.

Belts and safety lines are provided at many plants for work in cars where the ore is likely to slide and carry the men through the doors into deep bins or when the trestle is high above the ore pile. Similar devices should be used when it is necessary for a man to enter bins to clean them out, or when men must work in them regularly to keep the ore moving down to the doors. The use of belts and safety lines is stated to have prevented several serious accidents at a large plant in the Pittsburgh district, where they have saved men from being covered in a slide of ore, or have enabled men carried down in a slide and covered up to be found at once and dug out without delay.

A steel platform at ore-bin chutes that promotes the safety of men filling ore buckets is shown in Plate II, A. A trestle with grating between the walk and the rail girders is shown in Plate II, B.



A. STEEL PLATFORM AT ORE-BIN CHUTES FOR SAFETY OF MEN FILLING ORE BUCKETS.



B. BLAST-FURNACE TRESTLE, SHOWING GRATING BETWEEN WALK AND RAIL GIRDER.



FALLING AND FLYING OBJECTS.

1. Assistant foreman was on car to dislodge lump of ore frozen to side of car; lump fell and struck man, fracturing thigh. B.
2. Ore unloader was in car, standing in bottom, and barring ore down through doors. Lump rolled down from top of pile and struck him on leg. B.
3. Trestle man was passing coke car when bar laid on top of car side fell on he. d. B.
4. Unloader in coke car had dust blown into eyes. D.
5. Carpenter was tearing down old trestle when the structure collapsed and he was crushed by falling timber. A.

Falling of ore or limestone lumps is the main cause of injuries from falling material. Three other accidents similar to No. 1 and 11 other accidents similar to No. 2 occurred. The most hazardous condition is introduced when the carload is frozen and the ore or stone beneath the frozen crust is cut away. The work is especially dangerous because, in cold weather, foremen and men are apt to be benumbed and to lose their ordinary caution, and, in addition, the work is usually behind, the bins are low, and every consideration urges haste. In order to keep the furnace going chances are taken in unloading, as to stop the furnace at a time when it is working badly or when it is low might introduce a greater hazard than is faced in hurry-up work with frozen ore and flux.

At such times, every aid that will minimize the danger should be employed. The provision of a steam line, with hose and gas-pipe nipples to insert in the ore for four to six hours before the car is spotted on the trestle, is a great aid in unloading and is widely employed where the capacity of the ore yard is insufficient and ore must be shipped through the winter season; dynamiting the top crust and large lumps of frozen material is helpful; where natural gas is available at low rates it is employed with good results; in western Pennsylvania, the practice is common of keeping in touch with the Weather Bureau, and when a cold snap is imminent, shipments are stopped.

In using steam hose, care must be taken that the hose is in good condition, as it is apt to deteriorate under extreme temperature changes.

In using dynamite, only small charges should be used; usually half a stick is sufficient, and for best results the hole in the frozen crust should be at an angle of about 45°. Firing is most safely done with a firing battery, warning being given, and a man being stationed to see that no one comes within range of the flying pieces of ore. Dynamite should always be thawed before using, as the use of frozen dynamite is very dangerous. Standard thawers should be provided for thawing, and no attempt made to thaw the explosive by placing it near a stove or boiler, or in a pail with hot bricks.

Such accidents as No. 2 are to some degree unavoidable. Men standing in supposedly safe positions are sometimes struck by rebounding lumps of material. When it is necessary for men to get down into the car to pull the ore down in the hopper to the doors, the safest place to stand is between the two hopper doors, where there is the least chance of being struck by a lump of ore or by the collapse of the face of the pile.

HAND LABOR OR HAND TOOLS.

1. Two laborers were unloading heavy scrap and one allowed his end to slip. The piece in falling crushed other man's toes. C.
2. Scrap rolled from pile in car onto man's foot. D.
3. Laborer was throwing scrap from car when hand leather caught on rough edge of scrap, wrenching arm. D.
4. Laborer unloading lumber allowed piece to slip, tearing nail from finger. D.
5. Ore unloader was sledging bottom of car to make ore slide out of door, and caught wrist on cotter pin of winding rod. D.
6. Laborer strained muscles of abdomen in working with car wrench. D.
7. Laborer ruptured blood vessel in neck in lifting timber. D.
8. Laborer cut hand on rough scrap. D.
9. Laborer allowed lump of ore to drop on foot. D.
10. Unloader was opening door of limestone car, when door opened suddenly, causing wrench to jerk, and threw man into bin where he was covered by limestone from car. A.
11. Stock unloader was holding bar when man sledging on it hit him in back. D.
12. Bin man squeezed fingers between bar and ore-bin girder in loosening ore in bin. D.
13. Laborer struck kneecap against end of bar. C.
14. Ore unloader's pick struck shovel lying on cross strut in car, causing it to fall on finger. C.
15. Bin man jammed hand between end of bar and ore-bin girder in withdrawing bar from sticky ore. C.
16. Man placed pinch bar under car wheel to stop car; wheel catching bar threw it up against car frame, jamming hand. D.
17. Trestle man was holding steel wedge being driven in frozen ore, when steel sliver from wedge struck his eye. Loss of sight.
18. Sledge came off handle and struck man on head. C.
19. Coke unloader was beneath car working on warped door while helper was trying to close door; door dropped and hit unloader's back. D.
20. Laborer went beneath car to pry open door, when it opened suddenly, squeezing chest. D.
21. Laborer, in dropping car door, caught palm of hand in cotter pin at end of winding post. C.
22. Laborer scratched elbow and did not report injury, but dressed it with kerosene and tobacco. Infection. C.
23. Laborer pinched finger and did not report for five days. C.
24. Laborer fell from car and scratched palm of hand. Did not report injury. B.
25. Laborer's hands became blistered. Did not report. D.

The handling of scrap caused many accidents similar to the ones mentioned. Safety hand leathers would have prevented many of them, but a considerable number were caused by one of two men letting go a piece of scrap before the other. The handling of rough scrap is necessary, and such accidents, like most hand-labor accidents, are due primarily to a dulling of carefulness by the continued repetition of uninteresting work and its seemingly little danger. The prevention of such accidents is a difficult problem. Although supervision helps, there are limits to the extent to which it can be carried. The substitution of machinery for hand labor is a more effective means. For instance, one plant handling ladle and runner scrap from six furnaces has almost entirely eliminated accidents from this source by the use of magnets in the cast house and on the trestle.

Many of the types cited illustrate again the necessity of familiarizing everyone with the hazards incident to the use of hand tools and to hand labor. Personal caution by the men and close supervision by foremen must be insisted on if such injuries are to be reduced to the minimum. An inspection committee, an inspector of tools, or a foreman should examine all hand tools once a week to insure that they are in safe condition, to prevent accidents similar to Nos. 17 and 18.

Car wrenches cause many accidents around the trestle. The sudden dropping of doors with a weight of several tons of material on them will sometimes cause the wrench to fly about suddenly and may easily catch the wrench man in an unfavorable position. Twenty-three accidents similar to No. 10 occurred, two being fatal. The manner in which these accidents occur is varied. The man may not be in a position to detach the wrench from the winding post when it turns suddenly, or the motion may be so instantaneous that he has not time to loosen his grip on the handle. Consequently he is thrown against the car or walk or into the bin, or the wrench may fly about entirely, throw the man into its path, and deal a severe blow on his head or body. Sometimes excess of caution in avoiding these injuries leads to harm. Eight injuries were caused by the wrench slipping off the winding bar. The men are, not unnaturally, apprehensive of injury when the car doors are opened and are usually careful to so place the wrench on the winding bar that in event of its turning suddenly the wrench will be thrown off or may be detached easily. As a result of misjudgment in placing the wrench, or because of a rounded winding-bar nut, or a slippery wrench socket, the wrench may slip off the bar and cause the user to fall, or it may fall on his feet or catch his fingers.

Such accidents show the need of using safety car wrenches. These wrenches are already widely used, but some are not literally safe. Five accidents occurred with so-called safety wrenches. As an ex-

ample, a man was opening the car doors with a safety ratchet wrench which failed to work when the doors opened suddenly, the wrench flying around and causing a compound fracture of the great toe, with 42 days' lost time.

However, safety car wrenches should be used, as there are types that afford security. On no account should square-socket, nonsafety wrenches be used.

RAILROAD EQUIPMENT.

1. Stock unloader was on coke car loosening brake, and when the catch was kicked out the wheel whirled about so violently as to throw man off brake platform onto track. D.

2. Laborer was dropping two empty cars from trestle and could not manage brake. When they bumped the string of empties, he was thrown into car. D.

3. Foreman was dropping cars on trestle when brake would not work. He jumped from car, ran ahead to next car to set brake, and when cars bumped was thrown to rail, wheel passing over toe. C.

4. Brakeman was setting brake on car, and when it bumped was thrown off. D.

5. Stock unloader was caught between two cars. B.

6. Laborer was unloading car when it was bumped by a drag of cars being placed on the trestle. Man was thrown through drop doors to ore pile. D.

7. Laborer was run over and disemboweled by transfer car. A.

8. Laborer was in bin cleaning it when transfer-car doors opened just as it passed over bin, allowing contents to fall on man. D.

8. Stock unloader was spragging car with block of wood and wheel threw block in such a way as to crush finger. D.

9. Transfer-car man was burned by the short-circuiting of the light wire in the cab. D.

10. Unloader had finger pinched between car-door latch and ratchet. C.

Whenever men are working in bins they should be protected by track torpedoes or flags; they should not be allowed to enter bins over which larry cars or ore bridges with loaded buckets are running. In spragging cars, short pieces of lumber should not be used. Whenever cars are spotted on the trestle and the cars previously spotted are removed, the foreman should be sure that all men are out of the cars before the engine or new drag bumps the cars in coupling on or pushing them down the trestle. Cars in which men are working should be protected by a flag placed and removed by the foreman. A larry-car operator is primarily responsible for the safety of men who inadvertently get in the way of the car. Inasmuch as the car runs at irregular intervals, men can not, within reason, be expected to be continuously on the alert for it and do their work effectively. Especially when the car is running with the operator at the rear should he have the car under control and blow the warning whistle frequently, because a clear view of the track can not be had. The possibility of injury through lack of control of cars by reason of deficiency in skill or in the brake itself is evident, and although the

men engaged ranged from skilled to unskilled, it is always better to have men accustomed to the work do these jobs.

MISCELLANEOUS TRESTLE ACCIDENTS.

1. Laborer hung a torch under car while he was working on a car door to get it open. He bumped the torch, causing it to upset and spill burning oil over his arm. D.

2. Laborer opened doors on flue-dust car and hot dust blew out over his feet. D.

From time to time, throughout the entire works, the common torch or "smoke pct" either upsets or explodes when placed on a hot surface, causing more or less serious burns. There are many places where the use of an electric light on an extension cord is not feasible and where the torch seems necessary, but it should be so placed that it will not be struck by tools. Prevention of flue-dust accidents is discussed elsewhere (pp. 62-69).

To conclude the material on trestle accidents, mention may be made of the importance of having the trestle walk sufficiently wide so that men will not be in any danger of falling into the bin when they are thrown off balance by a car-wrench mishap. The walk can extend within 12 inches of the rail girder. Another safeguard is to have a grid composed of round iron bars between the girder and the walk. (See Pl. II, B.) Each pinch-bar handle should be provided with a steel disk, and the heel should not be allowed to get smooth.

STOCK HOUSE.

1. Helper on larry car jumped off while car was in motion and fell against wall. D.

2. Men in skip hole cleaning up left trapdoor open. Stock-house men walked into open hole, falling into skip pit. D.

3. Larry-car man, standing on top of scale car pulling coke from bin, fell to ground. D.

4. Boiler maker's helper was on temporary scaffold repairing bin chute. Board was insecurely fastened, causing him to fall to floor. B.

5. Boiler maker in repairing bin chute propped bin door open with piece of lumber. Was sitting in chute when prop slipped, letting bin door drop on leg, breaking bones above ankle. B.

6. Boiler maker was repairing ore-bin door when larry-car operator put feed mechanism in gear to draw ore from the bin. Boiler maker was thrown to floor. D.

7. Man placed handles of wheelbarrow on end of larry car and then sat down in barrow. Larry car started, upset the barrow, throwing man out onto track, and ran over him. A.

8. Man was thawing ore in chutes with gas. Turned on gas and attempted to light it with a match, when the gas flared out and burned him. D.

9. Laborer was working under empty bin; hot flue dust was dumped into bin and some escaped from door of bin and fell on him. D.

10. Barrow man was burned by hot sinter dumped into and escaping from door of nearly empty bin from which he was drawing sinter. D.

11. Bottom filler stepped into pail of scalding water. D.

12. Larry-car helper was throwing switch and was struck by larry car. C.

13. Laborer while cleaning tracks was bumped by larry car and thrown to ground. B.

14. Man sitting beneath bins with feet toward track slipped from seat as larry car passed. Foot went beneath wheel and toes were slightly crushed. D.

15. Repair man had leg broken when larry bumped idle larry upon which he was working. B.

16. Laborer cleaning stock house was struck by scale car and suffered an injured hip. B.

17. Clean-up man wheeling scrap to skip pit was caught between two larries. D.

18. Helper was cleaning ore from track at skip pit where larry car had just dumped charge. Operator, not seeing him, started car and caught foot. D.

19. Stock-house man was standing on skip car working on larry-car doors, which were stuck. As car doors opened, car moved and wheel caught man's fingers, they having been on rail to keep him from falling while on skip-car edge. C.

20. As man was shutting limestone chute, lever broke, causing him to lose balance and fall. D.

21. Nut on bin-chute mechanism fell on larry-car helper's head. D.

22. Larry-car helper caught hand between side of bin-door chute and door. D.

23. Bottom filler, in prying lump from bin chute, had fingers caught by lump when it slid out. C.

24. Larry-car helper standing on top of car side was struck on foot by large lump of ore he dislodged from bin door. C.

25. Larry-car man in operating limestone-bin door was struck on foot by lump of limestone. C.

26. Piece of limestone fell from chute on larry-car man's head while running car beneath chute. D.

27. Laborer was hit on foot by lump from poorly closed bin door. D.

28. Barrow man let piece of scantling used to block ore in chute fall on foot. D.

29. When scrap was dumped from wall into stock house, piece flew and hit man's face. D.

30. Scrapper, in pulling scrap to scale car, fell when hook slipped off scrap. D.

31. Scrapper cut hand on sharp scrap. D.

32. As man was throwing scrap into skip car, his glove caught, pulling him down into bucket. B.

33. Coke dust flew into man's eye. D.

34. Man let scrap fall on foot. D.

35. Helper loading heavy scrap into barrow had flesh on finger scraped to bone when fellow workman shoved the piece before it had cleared the top of barrow. B.

36. Scrap man pulled piece of scrap from bottom of pile, when pile tumbled, crushing man's toes. B.

37. Larry-car man's helper in throwing the switch put foot on lever to force it down, allowing other foot to remain beneath lever, and fractured great toe. C.

38. Man loading ore buggy from stock pile let lump fall on foot. B.

39. Barrow man in pulling barrow struck head against bin. D.
40. Man holding bar was struck on nose with sledge. D.
41. Bottom filler pulling buggy slipped and struck kneecap. C.
42. Helper pulling buggy had foot run over when one wheel struck coke and made buggy swerve. D.
43. Helper ran buggy wheel over fellow workman's toe. C.
44. Helper was pushing barrow when wheel hit lump on floor. Barrow swung, catching hand between handle and post. D.
45. Buggy was run into another, catching bottom filler's hand. B.
46. Barrow man, in pulling off scales buggy in between two others, caught and fractured finger. B.
47. Barrow man lifted end of barrow to try weight. Helper placed hands on back to push barrow away, when barrow man let end down catching helper's fingers between top of barrow and bin chute. B.
48. Barrow was run into helper pushing on another barrow. D.
49. Man allowed buggy to overbalance backwards, and barrow leg hit shin. D.
50. Barrow man let barrow leg down on heel. C.
51. Cager strained himself in moving heavy ore barrow. Did not report for work for 22 days, when he was found to have hernia as result of strain. B.

Accidents Nos. 1, 2, and 4 illustrate the dangerous practices of getting off cars in motion, failure to guard floor openings, and lack of inspection of scaffolds before use. It should be a part of a foreman's duties to correct such practices.

No. 3 illustrates a type of accidents of which nine similar accidents occurred, usually when the ore was sticky, lumpy, or frozen. At such times it is necessary for the larry-car men to do this work from the side of the car, standing on the edge of the hopper. They may be in positions where a rush of ore or a lump may hit them. Four accidents of a similar type occurred where men were working alongside of hand barrows. Care must be taken by men to stand away from the path of falling lumps, and whenever headroom permits platforms from which bar men can work should be provided. Some of the older types of bin doors make it necessary for men operating the doors to stand dangerously close to the path of the falling material. These conditions should be remedied.

Accident 5 illustrates the need of close supervision of methods of work that is essential to put work on an absolutely safe basis. Although it is impossible for a foreman to accompany each man to his work and personally supervise him, nevertheless the job, if unusual, should be examined by the foreman and explicit directions given, especially as men unaccustomed to the plant may easily encounter unfamiliar jobs for a considerable period. All bin-door mechanism should be guarded, and if some one is adjusting or oiling some part of it the larry-car operator should be notified so that he can warn those engaged in the work in case it becomes necessary to use the bin.

Accidents 7 and 8 illustrate thoughtlessness, an element that can be eliminated only through a strenuous campaign of education in safe ways of work. Gas should never be lighted at the end of nozzles with a match, as the control of gas flow is uncertain. Burning waste, coke, wood, etc., placed in the chute or on the floor, should be used.

Regarding accidents 9 and 10, before material of any description is dumped into an empty bin, the stock-house force should be notified so that they will be away from the door. Flue dust or flue-dust products should be thoroughly wet before being dumped into the bins.

In each of five accidents in which men were struck by scale cars the cars were equipped with continuously sounding gongs. The danger of such warnings is that the men become used to the continual noise by long association so that it ceases to be effective. The weight of evidence is against a continuously sounding gong. Much better is an emergency signal, preferably a loud whistle, operated by a foot button or lever, as the operator's hands are fully occupied with the controller and foot brake.

Accidents like Nos. 18 and 19 could, of course, not well be prevented by warning whistles. No. 18 would have been avoided if the helper had set the brake, and No. 19 if the stock-house man had kept out of the way after the car had been dumped.

Most of the accidents in handling scrap, of which there were many more than mentioned, were due to the lack of ordinary care and skill. It is difficult to distinguish between the two, because a man sometimes misjudges the weight or balance of a piece and it turns and slips from his grasp. Such accidents are not the results of carelessness but of error of judgment or of lack of experience.

The considerable number of hand-labor and hand-tool accidents encountered are not reviewed, as they do not differ from others reviewed elsewhere. The significant fact is that long employment gives no immunity from injury to men engaged in these tasks. The variety of tasks they do is almost infinite; few safeguards can be applied, and the obvious need is training.

In handling scrap the safety hand leather is absolutely essential, as is illustrated by accident 32. A safeguard for the handling of scrap being taken to the stock house is an inclosed chute, which prevents the pieces from breaking and flying when dropped.

In regard to buggy accidents, of which accidents 41 to 51 are types, buggies are in use at only two large plants, though several small one-furnace plants have this equipment. One large stock house with this equipment reported no buggy accidents resulting in disability in excess of 7 days, whereas another, equally large, had

several accidents causing disability in excess of 14 days. One plant has intensive personal safety work and the other does not, and therein lies the explanation. The work is perhaps semiskilled, as the ore buggy, loaded, weighs as much as 2,350 pounds. Not a little dexterity is required in balancing, pulling, and dumping it. If a man's hand or foot is caught there is certainty of injury of some degree.

However, accident prevention is a matter of skill and continued care and it is impracticable to particularize as to ways and means of attaining this skill in managing barrows. Foremen in stock houses should be on the lookout for growing carelessness, and should take every opportunity to show new men, as best they can, how to handle the barrows with the least labor and danger to themselves.

Accident 51 is typical of the repeated instances of reluctance of men to report injuries. There were two cases of infection in the stock houses; in one the man did not report his injury for 11 days, and a small scratch became a serious injury. In the other a man abraded the skin of his leg through the cloth of his working clothes, did not report for 5 days, and lost 20 days as a consequence of infection.

The feeling that prompts men to ignore little wounds is their experience that former slight scratches or cuts have not caused trouble. Added to this conviction of immunity is the reluctance to report the injury for fear that it will result in their being ordered to stop work for a few days. Loss of wages may be a serious misfortune to a man earning an average wage about a furnace plant. The experience of more than one large company in the last two years indicates that it is possible to shorten to a marked degree the lost time attendant on minor injuries. Efficient treatment of minor injuries prevents a serious loss of time for those injured, and consequently their reluctance to report their injuries disappears. The reduction of lost time attendant on slight injuries is one of the most important accomplishments to be achieved in getting men interested in safe methods of work.

SKIP PIT AND HOIST.

1. Cage being a little above the top platform, top filler started to lower it. Brake was on tight and he had to turn on much steam to start cage; then it dropped 5 feet before he could stop it. As cage at bottom started suddenly, man riding on it jumped off and was hit on head by safety gate. D.

2. Top filler started cage up as man was taking barrow from cage, and the bottom filler was thrown; contusion of head. D.

3. Stock-house man knelt down to open skip-pit siphon valve next to skip incline. Placed hand on skip-car rail and skip passed over hand. C.

4. Millwright was repairing steam insulation on steam line running up skip incline to top of furnace, and was struck by descending skip. C.

5. Motor inspector was repairing overtravel device and allowed tool to touch contact plates, causing flash. C.

6. Electrician working on rear of switchboard touched tool against a bus bar, causing burn of wrist. D.

7. Pipe fitter was standing on ladder taking down pipe when pipe wrench slipped, causing ladder to tilt. He jumped and injured side. B.

8. Piece of coke fell from skip, striking laborer on head. D.

9. Man was cutting rivets at skip hoist when one flew and struck laborer standing nearby. D.

10. Laborer was cleaning out skip pit when lump of limestone fell from bucket onto foot. D.

11. Millwright foreman strained back while changing cables on skip counterweight. D.

12. As buggy man was pulling barrow onto cage, the momentum of barrow pushed him against wall. D.

13. As barrow man was pulling buggy onto hoist he lost control of it and was jammed against the side of elevator. B.

14. Laborer in pipe gang was disconnecting pipe from skip-pit siphon when steam was turned on and scalded his legs. C.

15. Stock-house laborer was cleaning up along the stock house when material thrown by a furnace slip struck him. D.

16. Helper was oiling skip-car wheels when hoist man let car down unexpectedly. C.

Accidents similar to Nos. 4 and 16 can be eliminated by locking open the skip-motor switch before working on the skip or the skip incline.

Accidents similar to Nos. 1 and 2 may be avoided by not allowing men to ride on the hoist to and from the furnace top. In so doing they are constantly subject to accident from failure of overtravel devices, cables, and safety clutches, and from falling material. Such a rule, although difficult to enforce, and seemingly impracticable, is adhered to at certain plants. If men are allowed to ride on the hoist there should be provided screens at the sides, a cover over the top, and efficient safety catches. The cables should have especial attention as regards breaking strands and lubrication.

Three other accidents similar to No. 2 occurred and were the result of the custom of having the control of the elevator in the hands of the top filler. It is the rule that the top filler is not to move the cage until the cager at the bottom has signaled him, usually by a gong. As this signal is given hundreds of times a day, it may become so associated with routine that, even when not given, it is believed to have been sounded and the hoist is started. Unless constant care is taken by the top filler to see that men are off the hoist after placing the barrows, an occasional error in premature starting of the hoist may result seriously, especially when the hoist is equipped at the bottom with safety gates which may drop and catch men stepping from the hoist as it unexpectedly starts upward.

The equipment of hoists with safety gates at the bottom was prompted by the occurrence of accidents to the bottom fillers. As they stood at the foot of the elevator to take the barrows off the cage

as it came to rest, their feet would often project over the sill and be caught by the descending elevator. These gates are now standard equipment. A gate still in use at the tops of some elevators is of the trap-door type, which is lifted bodily by the elevator as it nears the top. These are dangerous and should be replaced with side-lift gates.

No one should be allowed on the hoist after the cager has given the starting signal.

Accidents 5 and 6 illustrate the necessity of standing on a dry board or rubber mat, of wearing rubber or dry leather gloves, and of having tools thoroughly insulated with fresh tape, when working on mill circuits even of low voltage. These circuits are sometimes grounded, and especial precautions are called for.

Accidents like No. 7 demonstrate the need of safety hooks at the tops of portable ladders, as well as of spurs or nonslip pads at the bottom. If hooks are not provided, or can not be utilized, a helper should hold the ladder. If a man must use a pipe wrench while standing on a ladder, he should be especially careful to avoid positions that invite falls should the wrench slip.

Accidents like No. 10 are infrequent, as most plants have a wide pit so that men have opportunity to retreat from beneath the rubbish bucket as it is hoisted and dumped. No accidents are reported as the result of falls of men in getting into or out of pits, even though there are some pits where a considerable degree of agility is required. Neither were there any accidents reported to men while in the pit caused by being struck by skip cars, either by movement of the skip or the breakage of the cables. Safeguards are the provision of a wide and deep skip pit so that men can get out of range of the movements of the car, and the requirement that the skip cars shall be empty while the men are in the pit cleaning it. At a few plants the skip car is chained when men are in the pit.

Accidents similar to Nos. 3, 8, and 9 have suggested the following safeguards, which are widely employed: The use of a sheet-iron guard outside the skip-track rails; covering the bottom of skip hoists with sheet iron (see Pl. III, *A* and *B*), and the use either of a shield to stop flying rivets or the provision of a warning sign in the path of the flying material.

Accidents 12 and 13 happened to men with 5 and 10 years' experience, and illustrate the danger inherent in the loss of control of heavily loaded ore barrows. Several superintendents have stated that when a furnace is started with a comparatively new crew in the stock house, they always experience a series of accidents with hand barrows, the number decreasing as the men acquire deftness. In telling new men how to be cautious, accidents similar to these are of use as illustrations.

Accidents like No. 14 occur rarely but persistently and are usually serious. The only effective method of preventing burns from a steam exhaust or similar line is the provision of danger tags or locks to be placed at a steam valve when work is to be done on the line beyond the valve.

FURNACE TOP.

1. Laborer, with three other men under direction of foreman, was cleaning furnace top and was severely asphyxiated by gas escaping from about the try hole. D.

2. Larryman's helper was helping remove a piece of scrap that was stuck between the bell rod and the hopper throat. Was overcome with gas and fell from top of receiving hopper to top platform. C.

3. Stock-house foreman had opened manhole on gas seal over big bell and was looking in to observe distribution of ore on bell when gas ignited and exploded, burning face. D.

4. Larryman's helper was cleaning top platform. Cleanings were shoveled in through gas-seal door and gas exploded inside and blew them out of the door over man's face. D.

5. Three men were sent to top of furnace to remove scrap caught between little bell and its seat. Door on gas seal was opened to loosen scrap, and as the door was opened gas exploded. D.

6. Furnace was shut down for changing cooler. Water-seal valves were filled, isolating furnace, and stock-house men were sent on top to clean up about platform and explosion doors. An explosion occurred in the furnace above the stock line and blew one man from an explosion-door platform to the ground. A.

7. As rigging gang was placing new plate in receiving hopper, man fell from top of the hopper to the furnace-top platform. A.

8. Rigger was carrying chain up ladder to platform over hopper when he slipped and fell. D.

9. Man employed in repairing top rigging had scale blow in eye. Scale was removed by fellow workman and infection resulted. B.

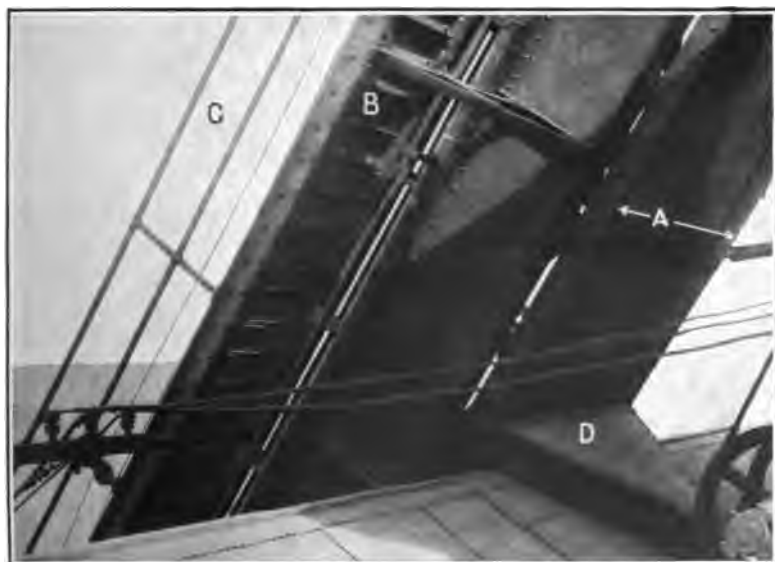
10. Sweeper dumped barrow into hopper and, in pulling it back, barrow swung and scraped hand against side post. C.

The furnace top, in the past notoriously dangerous because of escape of gas, has become less dangerous on account of the tight construction now in vogue. To men working about the top platform, the only menace from escaping gas is usually from gas escaping from the try hole. It is the practice at a few plants to keep on the furnace top a quantity of asbestos rope with which to pack the try hole when men are cleaning or examining the top platform sheave wheels, rodding holes, or operating cylinders. This simple safeguard is efficient and is to be recommended.

When it becomes necessary to work about the receiving hopper or on the sheave-wheel platform on top of the receiving hoppers, men are not uncommonly exposed to gas. In 1914 three accidents from this cause occurred. In one, while a millwright was repairing the telltale on the sheave-wheel platform, he was overcome, and



A. SKIP INCLINE COVERED ON BOTTOM; PERMANENT PLATFORM FOR SPRAY IN SHELL.



B. SKIP INCLINE ADEQUATELY PROTECTED.

A, Steel-plate sheathing to prevent material from skip cars from falling on workmen passing below; B, stairway for workmen making repairs; C, handrail; D, chute for collecting falling material.

his helper started to assist him down the ladder to the top platform. Both fell about 18 feet. The helper lost 2 days and the millwright 40 days. Another man, while oiling the sheave wheels over the receiving hopper, was overcome and fell in attempting to descend, receiving a fractured skull and rib, necessitating a long layoff.

On account of the imperfectly tight seal of both big and little bells against their seats, work of the above description should not be undertaken until the stock-house foreman has been notified and until a round of ore has been charged onto the big bell, with the last skip load, preferably of fine ore, on the little bell, to seal them as much as possible. The bells should not be operated while the men are on top. Two men should be assigned to even the most trivial routine or occasional job on the furnace top, as it is impossible to foresee dangers from escaping gas. A millwright going alone to the top of a modern furnace, in direct contravention of a long-standing rule, was found fatally asphyxiated by his helper, who followed him a few minutes later. This accident occurred in 1915 in another State and indicates the imperative necessity for strict adherence to the practice of having two men on such work.

For the use of men on top, who are in danger from gas, there should be installed a signaling device, preferably an electric gong placed in the cast house and operated from the top only. One plant has a good practice of requiring a brief signal to be given every 10 minutes, the omission of the signal being notification to the stove tender to investigate. At a few plants, for work around tops where excessive quantities of gas escape, or for work for a few minutes in a locally gaseous atmosphere about or over the hopper, half-hour oxygen breathing apparatus are used with success.

When it is necessary to examine the big bell through a gas-seal door while the furnace is in operation, or if it is the practice to shovel top cleanings into the big-bell hopper through a gas-seal door, the gas should be lighted by throwing in a bunch of burning oily waste. This should be thrown from a distance of at least 6 feet or should be put in with a long rod. When the gas has ignited, a gas-seal door on the opposite side should be opened and the little bell lowered. The gas should be kept burning by keeping a bunch of oily waste continually burning inside the hopper. This practice prevents an accumulation of gas and air in explosive proportions. Such mixtures explode violently even in the absence of any flame and at temperatures usually considered below the ignition point. Speculation on the cause leads nowhere; the condition is well known, and accident is prevented by keeping the gas burning inside the gas seal.

Opening gas-seal doors, which must be done before the gas can be lighted, is not usually dangerous, provided it is done quickly and the men get away immediately. The little bell should be kept shut to

prevent an induced draft of air into the gas seal. When the little bell is wedged open by a piece of scrap, the work must be approached with caution, as the gas is apt to be unduly hot on account of enforced delay in charging, and on that account is more apt to explode when the gas-seal door is opened and air is drawn in by the open little bell. The most effective safeguard against an explosion under these conditions is a $\frac{3}{4}$ -inch or $\frac{1}{2}$ -inch steam connection to the gas seal. The seal can then be filled with steam before the doors are opened. However, when the little bell is blocked with scrap, it is safest to take the blast off from the furnace, draft the gas back through a stove, and turn steam into the dust catcher, the furnace being isolated by water-seal valves from others in operation; a single furnace should have all burners shut. The big bell is kept shut, the little bell being open. The gas-seal doors can be opened within a few minutes with no danger either of explosion or asphyxiation. The vital point is a steam connection of adequate size. Three inches may be fixed as standard; there are a few 4-inch connections. J. W. Dougherty is entitled to credit for the impetus and publicity given this practice.

The accident resulting fatally shows that gas explosions at the top may be of considerable violence. That they usually are not is beside the mark. Preparations are necessary always to prevent the explosion assuming serious proportions. Even with the furnace drafted back there is usually a somewhat vigorous generation of gas within the furnace, sometimes sufficient to give a strong flame at the bleeders or burner if one or two are left turned on. Consequently, there is likely to be an accumulation of gas above the stock line, or top of the burden, in the furnace. When the furnace top is opened, either at the gas seal and bells or at the bleeders and explosion doors, and air enters, there is every probability of a sharp explosion. Even when a door on the gas-main system remote from the furnace top is opened and air is drawn up to the top by draft through an open bleeder an explosion is probable.

When a furnace is shut down for changing a plate, cooler, or tuyère which have been leaking badly, work on top is extremely dangerous, unless the most stringent precautions are taken. While the furnace is thus shut down the hydrogen content of the furnace gas will be higher. Instead of being 2 to 6 per cent, it may be as high as 15 per cent, and has been found to be more than 30 per cent. The hydrogen makes the furnace gas unusually explosive, and the gas may be "wild"; that is, coming off in unduly large volume. As a general rule, work about hoppers, bleeders, and explosion doors should be left until later unless the work is imperative. Cleaning the top can not be said to be in this class and is better left for some other time. If the work must be done, the bells and the explosion doors

should be opened and the gas on top of the stock line should be lighted with burning waste, either dumped in by the skip or thrown in by hand; the gas should be kept burning, and some one should be stationed to see that it does burn, and to keep a fire of waste going, throwing in the waste through the bell. The alternative is the use of steam.

To provide security for men replacing wearing plates or removing obstructions in the hopper, a permanent platform should be provided. It is provided at some plants, but is far from being standard equipment.

WORK INSIDE FURNACE AND ON FURNACE SHELL.

The work inside the furnace stack is confined to construction and relining, and accidents for the most part occur in connection with work on scaffolding, handling brick, rigging, and salamander removal.

1. As laborer was loading brick from pile of brickbats pulled from inside furnace, large tile rolled onto foot. C.
2. Laborer was picking up brick from pile when one rolled down on hand. D.
3. Laborer was piling brick inside furnace and caught fingers between two heavy shapes. D.
4. Brick fell from tub while it was being hoisted and hit laborer on head. D.
5. As laborer was piling brick in bucket, brick fell off working platform above onto arm. C.
6. Laborer's hand caught between balls of tubs as he was guiding empty to one side. D.
7. As bricklayer was cutting brick, fellow workman bumped elbow, causing him to cut palm of hand. C.
8. Small spall of brick being cut by workman embedded itself in eyeball of fellow workman near by. D.
9. While laying brick, man reached for brick just as helper put down armful; hand bruised. D.
10. Laborer was tearing lining out of bleeder, standing on lining inside pipe as he dislodged the brickwork. He missed his footing and fell through the bleeder, bruising leg, body, and face. D.
11. Man on scaffold outside of furnace shell fell when scaffold broke, falling about 24 feet. B.
12. While riveting from scaffold, rigger walked to end of board which was not fastened down. Board tilted, causing man to fall. A.
13. Five men were working on stack scaffold when it collapsed, throwing two men to the ground. The others grasped the top of the plate work on which they were working when they felt the scaffold going. A severe storm had weakened the scaffold on which 14 men had previously worked, and its condition was not observed. One A and one B.
14. Two men lowering steel plate received injuries when the tackle broke. Both D.
15. Man was holding bar for another to strike and was hit in chest, on account of standing on the same side as the hammer man. D.

16. Helper was digging out rubbish and brick inside hearth and was overcome with heat. D.

17. Hand of man holding drill for drilling hole in salamander became infected from contusions and blisters. D.

18. Rigger was standing on plank drilling hole for dynamite in salamander, when plank turned, letting foot down into hot water. In getting out, put other foot in. C.

19. Man was struck by piece of iron flying from blast in a salamander. D.

For pulling brick from the furnace, men should be given long hoes to enable them to stand away from bricks tumbling and rolling down the face of the pile. When men are loading and wheeling brick away from a pile, the "straw" or "shift" boss in charge at the bottom should have the men keep the brick pulled down to prevent a steep slope on the face of the pile.

The falling of bricks from hoist tubs and down through shafts from the working platform can be prevented by care in loading, or by using a tub with a cover attached to the bail, the cover closing automatically when the hoist is made. The working platform should be provided with trapdoors over the bucket shaft, and also over the ladder shaft, to prevent the falling of brick. (See Pl. IV, A.)

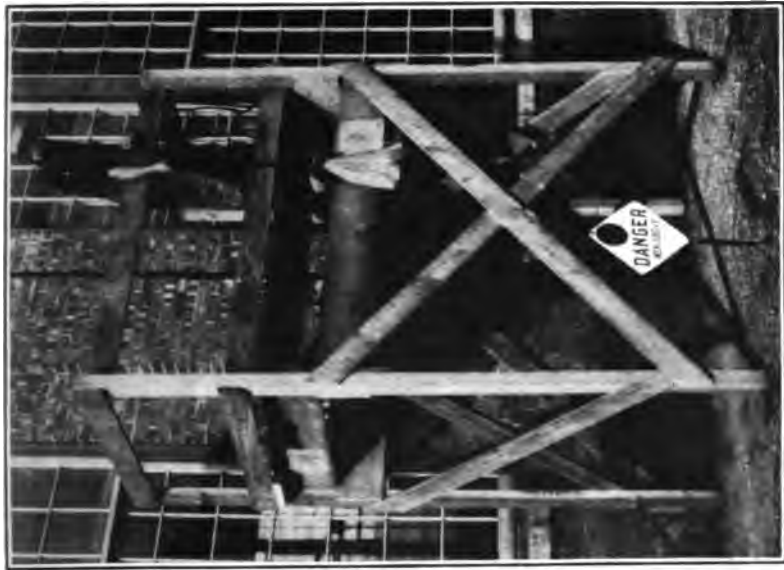
Injuries occurring from handling material can be eliminated only by not rushing the work too fast, and by carefulness in taking brick from the chutes and in laying them on the pile or bucket. In laying brick, the usual hazard is injury to the eye from flying pieces caused by chipping or cutting the brick to shape. Many plants now furnish goggles to bricklayers if cutting is necessary. Other injuries are caused by accidental hitting of the hand with the mason's hammer or by having the hand caught between brick in handling them.

Scaffolds (Pl. IV, B) should be erected and inspected with great care. The injury from a fall is often out of all proportion to the height of the scaffold. Thus two men fell from a height of only 13 feet when a hemlock board broke inside a furnace. One man had both ankles broken, entailing a disability of 104 days; his companion lost 11 days from concussion and laceration of scalp.

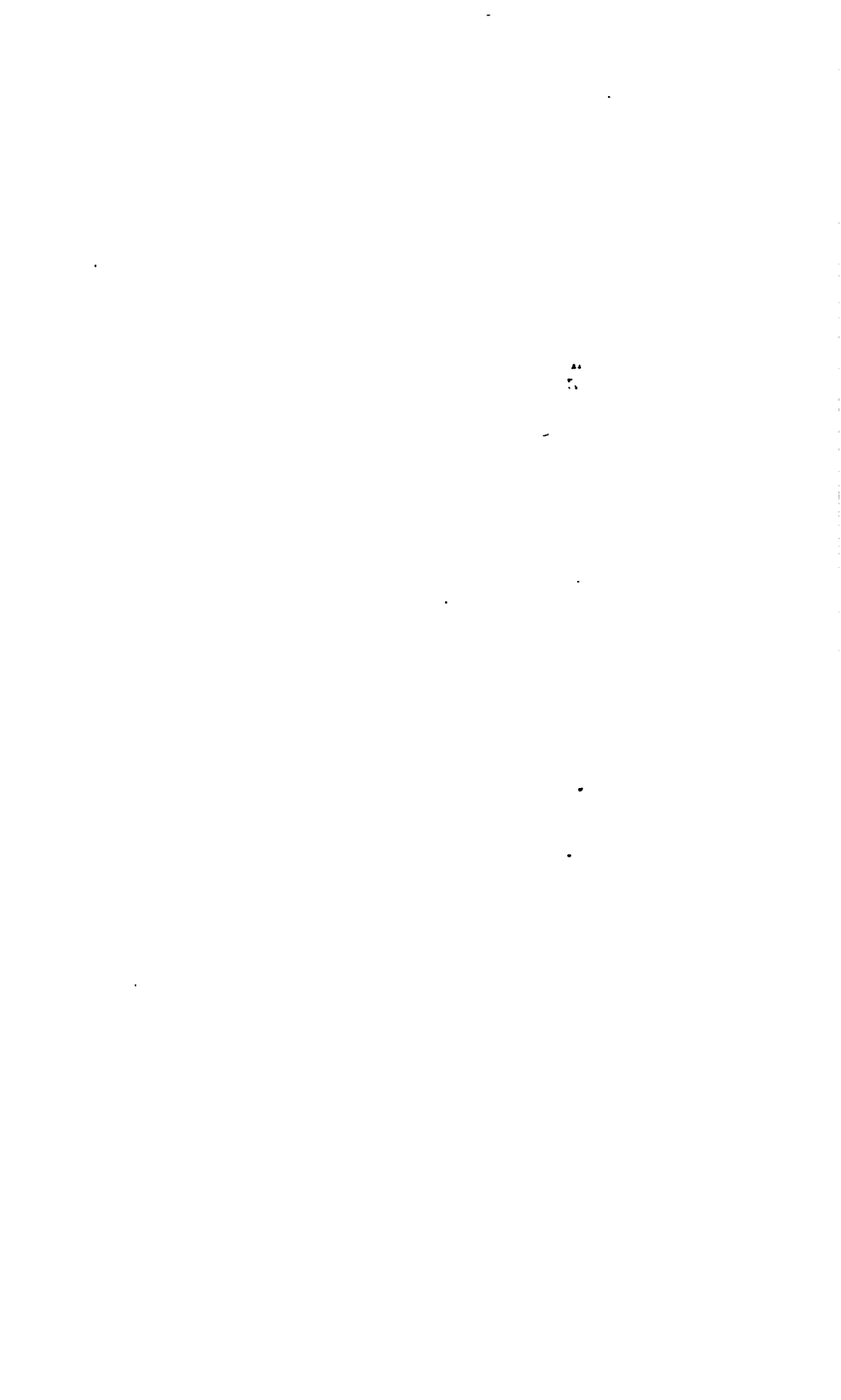
The construction of temporary and emergency scaffolds is sometimes left to millwrights, riggers, or handymen, and any material available is hastily utilized. This practice leads to makeshift scaffolding, weak in uprights, and with knotty, cracked flooring. Only experienced and reliable men should be employed in the construction of scaffolds, and the master mechanic should delegate to some competent carpenter the duty of inspecting scaffolds before they are used. The most desirable lumber is long-leaf yellow pine. It should be free from cross-grain, knots in groups, or decay. When the span is 12 feet or less, planks should be not less than 12 inches wide by 1½ inches thick. For a span that is between 12 and 20 feet, planks not less than 12 inches wide by 2½ inches thick should be used. When



4. SAFETY ARRANGEMENT FOR ELEVATOR USED IN RELINING BLAST FURNACES. RISING ELEVATOR LIFTS TRAPDOOR COVERING SHAFT. PINS PUSHED THROUGH STRAPS IN ELEVATOR FLOOR PREVENT LOWERING OF ELEVATOR WHILE MEN ARE ON IT.



B. SAFEST WAY TO WORK ON SIDE OF BUILDING. ERECT A STANDARD SCAFFOLD, AND USE PROPER DANGER SIGN.



more than two men are to work on the scaffold, a center support should be provided every 8 feet. The planking should cover the entire span between supports, and should not project more than 12 inches at each end of the span. Preferably, planks should be nailed down; otherwise, to prevent the plank from slipping sideways, a hole should be bored at each end of the plank and into the support, and a bolt placed in each hole. Scaffolds more than 6 feet from the ground should have a railing, which should inclose the three sides, away from the working side, and be directly over the supports at the extreme ends so that men can not walk out too far and tilt the plank up. The uprights should be 2 by 4 inches for scaffolds less than 25 feet high, and 4 by 4 inches if 25 to 50 feet high, and if repairs or construction work is to be carried on from the scaffold and if brick or metal work is to be placed on it, a table of safe loads should be consulted and the scaffold made as much stronger as required.

Swing scaffolds are preferably suspended by wire rope, which should not be less than one-half inch in diameter. The eye of the cable should be passed about a thimble and be secured by not less than two rope clamps. Needle beams, if of wood, should be at least 4 by 6 inches, or, if of pipe, the pipe should be 4 inches in diameter. Eyebolts through wooden needle beams should have the nuts secured by a split pin, or provision should be made to prevent the ropes slipping off the ends of either wooden or pipe beams.

Before men go on the scaffold to work, it should be tested for strength at twice the load it is to carry, and a man should be appointed to inspect it at frequent intervals. The men working on the scaffold should also inspect it and satisfy themselves that it is safe, but as many workmen are incompetent to judge as to its safety, the primary responsibility should be with the master mechanic or man to whom the duty of supervision is delegated.

Work on scaffolds should be carried out deliberately enough so that false steps are avoided. Loads should be brought down gently, not suddenly or heavily, and no heavy castings should be leaned against the uprights.

In 1914 one man was killed and several others were injured while springing holes in a salamander. This work consists in enlarging the bottom of a row of 1-inch or 1½-inch holes, previously drilled in the metal, by setting off in them constantly increased charges of dynamite, starting with a half stick and working up slowly to two or three sticks per hole until a crack develops along the row of holes. Then a heavy charge is placed in the holes and fired, the object being to split the salamander along an axis. This work is dangerous if the holes are not thoroughly flushed with water between each charge in order to dissipate any heat imparted to the walls by the explosion of the preceding charge. It is also dangerous

if the salamander is not thoroughly cooled off before the dynamite is charged. At temperatures above 175° F., dynamite becomes acutely sensitive to shock or to spontaneous ignition, and a great risk is taken when dynamite is charged in a salamander above that temperature. Before the hole is charged, the temperature should be determined by lowering a metal-inclosed thermometer into it; if "springing" or "chambering" is practiced, the hole should be cooled with water and its temperature should be determined with a thermometer at frequent intervals. The temperature should not be determined by putting a wooden stick or pocket rule down into the hole to see whether it chars, because the temperature at which wood chars is too high for safety. When the holes become persistently hot, the only even moderately safe practice is to roll each cartridge in a sheet of thick asbestos paper, with a wad of loose asbestos at the bottom, and to lower it thus insulated.

If dynamite is charged loose and stemming is employed, it is absolutely essential to use a wooden tamping bar. Although metal bars are sometimes used in earth, they will inevitably set off charges if they are used in metal-inclosed holes.

Shot firing is almost universally done with a firing machine. Precaution should be taken not to use a greater number of detonators than the rated capacity of the machine. The detonators should be connected in series, and the ends of the lead wires should remain disconnected from the machine and in the hands of a responsible assistant until the foreman has made all connections to the charges in the salamander, has given warning, has stationed guards, and has reached the detonating machine. The lead wires should always be carefully examined each time. It is better to get new wires than to twist broken ends together or to patch defective insulation, for any unnecessary resistance or grounding in the wires makes the chance of misfire more probable. A misfired cartridge or charge in a salamander is probably one of the most dangerous misfires with which one can contend, because the heat in the salamander is apt to set it off after the danger is thought to be past, and while attempt is made to remove it or to set it off again by digging down to it through the stemming, or by drilling a new hole near it. Some plants use the electric-light circuit with success to set off charges.

Both on account of the danger to furnace columns, and to life and limb, the hearth should be inclosed (Pl. V, A). Men should be stationed at every approach to the cast house, but out of range of flying pieces, to prevent any unwary employees from approaching. A curious employee wandered into a central Pennsylvania cast house just as a blast was set off. A flying fragment of salamander took off the seat of his trousers, causing no injury except considerable shock.



A. HANGING BULKHEADS MADE OF RAILROAD TIES, COMPLETELY INCLOSING HEARTH JACKET DURING BLASTING OF SALAMANDER; WOODEN PROTECTION ON FURNACE COLUMNS.



B. CAST HOUSE AND FURNACE BOSH.

Note grates over casting holes, railed walk on bustle pipe, shield protecting snort-valve lever, furnace pressure gage, and engine-room signal-whistle switch. Telltale light can be seen just under the gage.

In the past several fatalities have occurred in blasting salamanders, and it has become the policy of several companies to employ contractors specializing in such blasting, who do the work more quickly and efficiently and also more safely than is possible by members of the furnace force who only infrequently encounter the job.

THE FURNACE FRONT.

BUSTLE PIPE AND TUYÈRES.

Eleven of the total of 30 accidents reported from about the base of the furnace resulted from hand labor or the use of hand tools. However, the work was peculiarly characteristic of furnace duties and the circumstances should be of interest.

1. Tuyère man was cleaning and testing bosh plate and let it fall on foot. D.
2. Boiler maker's helper was placing bosh plate in furnace with block and tackle when the hitch slipped, causing him to fall. D.
3. Laborer on scaffold pulling plate allowed plate to fall on foot. C.
4. Handy man sat down on bustle pipe to pull on wrench, which slipped, causing him to fall off bustle pipe. B.
5. Hot-blast man was tightening bridle spring when wrench slipped off nut. D.
6. Larry man was helping change blowpipe, and pipe fell on his foot. D.
7. Keeper was using bar to guide blowpipe into tuyère seat and bar slipped, spraining wrist. D.
8. Keeper was holding bar to cut iron from tuyère while helper sledged. Helper missed bar and sledge landed on great toe of keeper. C.
9. Bottom filler was prying cinder from inside cooler, when a heavy piece rolled out onto hands. D.

With the exception of the fall from the bustle pipe, which occurred when the pipe was equipped with only a single hand rail, without intermediate rail, toeboard, or platform, the accidents are typical of the hazard of the work in changing cooling equipment and cleaning up messes or plugged tuyères. Cooling equipment is heavy, varying from the 110-pound tuyère or the 165-pound cooling plate up to a 600-pound cooler, so that being caught beneath by the appliances is apt to cause serious injury. Sledging, bar work, and wrenches also contribute a share of injuries. There is a better opportunity for supervision of awkward or unskillful methods of work about the furnace front, where work is almost always done under the supervision of a blower foreman, than at the trestle or stock house, where men are scattered along the entire length doing routine or minor emergency jobs away from the eye of any foreman.

The remaining accidents are typical of furnace hazards.

10. Hot-blast man was blowing dirt from eyesight, and as he lifted the eyesight hot dust blew under goggle shield, burning eyelid. D.
11. As helper was picking out tuyère and blowpipe with long bar through eyesight hole, cinder blew out, burning neck, face, and hand. D.

12. Blower was looking in eyesight when the glass blew out, blowing dust and hot glass in eye. C.

13. Tuyère man was loosening nut on bridle rod before crew put bar in eyesight to hold the tuyère stock up against the pipe. Blowpipe fell and gas blew out, burning face and ear. D.

14. Helper had hold of crossbar to help take down blowpipe when tuyère man turned water in tuyère. Escaping steam and hot water from discharge scalded back. D.

15. Helper was in the act of removing a discharge pipe of cooler in which the water had been turned off when some one opened the feed and escaping steam scalded abdomen. B.

16. Blower was claying up tuyère and the gas pressure was so strong that it blew hot coke from the opening, a piece falling into top of man's shoe. D.

17. As stove tender was helping plug tuyère gas blew out, burning face and arm. D.

18. Keeper was cleaning out cooler seat before inserting tuyère when an explosion of gas at the top caused flame and hot coke to shoot out of the tuyère opening. C.

19. As blower was taking the blast from furnace, cinder came back and burned through blowpipe, and as he ran to lever to throw mud-gun clamp down, cinder ran near him and one of crew turned hose on it to chill it. Steam scalded ankle. D.

20. Second helper was lying on bench under blowpipe when cinder came back and burned through the pipe, burning neck. C.

21. Tuyère had been turned down on account of leak. First helper was getting mud gun ready for cast, standing near tuyère, when iron suddenly came back in pipe and burned it, and flame and molten iron blew over man. B.

22. Helper was playing water on blowpipe and tuyère in which the iron was coming back. Blowpipe and tuyère burned and the molten iron exploded in contact with the water, blowing tuyère out and burning helper on shoulder, arm, and back. B.

23. Furnace millwright and hot-blast man were on scaffold at bosh arranging to change cooling plate. The blast was on and the plate blew out on account of loosening brace. Both men were thrown from scaffold, one fracturing collar bone and other receiving burns. Both B.

Accident 10 could have been prevented by providing the eye-sight cap with a cable and pulley, by which the eye-sight cap could have been lifted from a safe distance, or by the use of the stopcock type of eye sight with long levers.

Accident 11 would perhaps have been avoided if the picking bar for tuyères and blowpipes had had a 12-inch or 15-inch bar forged on the end at right angles, as is done at several works.

Accident 12 could have been prevented had the blower been wearing goggles or been using a hand glass between the eye sight and his eye.

In regard to accident 13, beyond giving the bridle nut not more than two turns to loosen it, the nut should not be turned back until at least two of the crew have put their weight on a bar inserted into the eye sight. The bridle nut should not be turned back at all until the blast is taken off the furnace and the gas drawn back or the

relief valve opened. In 1914 a blowpipe was dropped before the blast was off and the workman was severely burned by burning gas shooting out over him. He suffered a disability of 20 days.

Changing tuyères is a hurry-up job. Every three minutes the furnace is off means a loss of, say, a ton of pig iron. Crews are frequently in great rivalry, and the average time for this job is carefully kept, and not infrequently is posted. The danger of too much haste is self-evident.

Sometimes, when the nose of the tuyère is burned away and the tuyère is leaking badly, the water inlet is closed and water is played on the outside face of the tuyère to hold it until the furnace can be flushed and the blast taken off. Usually when the blowpipe is dropped the tuyère is red-hot and wedged tightly in the cooler by expansion, and it is usual to turn on the inlet momentarily to chill the copper seat. That it is essential to wait until everyone is away from the front of the tuyère is evident from accidents 14 and 15.

The use of leggings or simply tight-fitting shoes would have prevented the injury in connection with accident 16.

Occasionally the wildness of the gas, or the back pressure from other furnaces on the line, causes a long, intensely hot flame of gas to be ejected from the tuyère. At such times the work of claying back the tuyère is most difficult, as the flame may be longer than the stopping hook, and the force of the flame ejects balls of clay thrown into the tuyère. At such times the water seal at the dust catcher should be shut before the tuyère is clayed back. Many plants do this regularly on a tuyère change, whether trouble is anticipated or not. If the gas is still very wild at the tuyère and drafting back through two stoves does not relieve it, opening the top usually relieves the pressure. However, this expedient is not good practice, because it heats up the top rigging and sheave wheels and brasses, and may weaken the skip cables. Furthermore, if the furnace is isolated, it may cause a "top shot," or explosion of gas.

Accident 18 illustrates the danger of keeping the top open while changing tuyères. Such an accident could happen also with burners or dust-catcher bell open, drafting air up through the downcomer.

The safest practice in changing tuyères is to isolate the furnace with a water seal, draft back the gas, turn steam into the dust-catcher, and, if the gas is wild, open the bleeder. This procedure obviates to the greatest degree any chance of a top shot while working at the tuyère. Instructions must be given the bell operator not to lower the bells while work is being done, as the falling material from the bell may cause sufficient jar to eject gas or coke from the tuyère, or a top explosion may occur. Care must be taken to see that the furnace has settled, or slipped if it has been hanging, before the blow-

pipe is dropped. An accident in which several men were killed occurred many years ago when the furnace slipped while men were working about the opened tuyères. To facilitate knowledge of the condition of the furnace charge—whether “hanging,” “slipped,” or “down”—it is advisable to install in the cast house a signal light operated by the skip man. The simplest is a red light, signifying “hanging,” and white light, signifying “down” or “slipped.”

Another precaution against ejection of flame and red-hot coke from tuyère openings while men are working at them is to be sure that the crew have the tapping hole nearly stopped, if the tuyère is being changed after a cast. Especially hazardous is the use of water or sloppy clay for stopping the tapping hole while a tuyère is being changed. In 1914 a man was burned about the face while claying back a tuyère because wet clay was used in stopping the tapping hole. The wet clay caused an interior explosion of hot iron which threw a jet of flame from the tuyère. Some years previously a man was fatally burned in this manner in the Pittsburgh district. It is safest not to change tuyères until the tapping hole is stopped.

Before the tuyère is pulled, the blower or keeper should be as sure as he can be, judging from his experience, that the coke at the face of the tuyère has been tightly and heavily clayed back. It is especially necessary to make a good job when cinder is lying in the furnace. Cinder is sometimes present even when none appears at the monkey, and it runs out from the tuyère abundantly when the attempt is made to clay back.

It is safer for every one who does not have to get close to the work, to stay on one side while a tuyère is being changed. However, a good job of claying back is absolutely essential.

The danger from burning blowpipes and bursting tuyères is difficult to control. The only precaution is to stand to one side as much as possible when playing a hose on a tuyère or blowpipe that threatens to burn through, and never to stand in front of a tuyère unless it is necessary. Signals to the stock house and the blowing room, as well as snort-valve levers and electric-burner connections should always be placed out of range if possible and should be protected by a shield (Pl. V, *B*) if they can not be put in a safe place. Emergency signals should be placed outside of or on opposite sides of the cast house for use when the signal regularly used is unapproachable on account of flying coke and flame from a burned blowpipe or bursting tuyère.

Analogous to the bad practice of loosening bridle rods before the blast is off is the practice of removing braces or loosening bosh plates, preparatory to changing them, before the blast is off, illustrated in accident 23.

Bosh plates should be secured in place by braces, when they are set in plate boxes, to prevent their ejection. The braces should not be removed until the blast is off the furnace. Plates set in brick arches are not usually braced or otherwise tied in, the pinching and cementing effect of the surrounding brickwork and clay packing being sufficient to hold them in place. Such plates should not be started with jackscrews or other pullers until the blast is off. Blowers should see that such expedients are not used on any pretext of hurrying the work.

CINDER NOTCH.

1. Keeper was using tapping bar to open monkey when cinder flew and burned foot. Was wearing laced shoes. D.

2. As keeper was opening monkey, iron came over with the cinder, burning monkey and exploding in contact with water. Hot metal flew over keeper's neck and body. D.

3. Helper was using picking rod to remove obstruction from monkey while flushing cinder. In removing rod, cinder flew into shoe. D.

4. As monkey boss was botting up, cinder splashed onto hand. C.

5. Keeper was botting monkey and at bot entered monkey explosion occurred, throwing slag over keeper's face. D.

6. As cinder snapper was botting monkey, helper held shield too far toward front of notch; cinder splashed onto cinder snapper's shirt, setting it on fire. D.

7. As monkey boss was botting up he fell and fractured his leg. B.

8. Pipe fitter was cleaning seat of monkey cooler when gas and coke blew out, burning his face and hand. D.

9. Man was passing wood through cinder notch to fill furnace before blowing in when piece jammed and struck eye. D.

10. Monkey boss allowed bot to fall on foot. C.

11. Cinder snapper received fracture of instep by "welshman" falling off tapping bar. B.

Opening the cinder notch is perhaps the least hazardous of work about the front. Small splashes of slag are frequently thrown out but rarely come in contact with the person. At one plant a large shield is employed. It is carried by two arms pivoted on an adjacent furnace column. The shield is used when the notch is opened or closed, the tapping bar or picking rod being placed in the cinder notch through a hole in the shield. Small shields, from 15 inches square up, are also used when the cinder notch is opened to prevent small splashes of cinder reaching the man working alongside the fall or runner. (See Pl. VI, A.)

The same type of accident as Nos. 4, 5, and 6, with little variation, occurred at other plants, involving a cinder snapper, a monkey boss, and a keeper, with experience of 2 to 14 years. The injuries caused loss of time of 8 to 16 days, being burns of wrist, forearm, face, and feet. Such injuries are most effectively prevented by having men wear welder's masks—a shield of fine, strong wire gauze tied about

the hat, and extending down to the chest or lower neck and back to the ears—gauntlet gloves, woolen or hard jean cloth clothing, and tight shoes. The arms should not be exposed. It is helpful to have a plate shield alongside of the cinder fall, the end next the notch being far enough away so that a picking rod or bot can be manipulated.

Most of the 11 accidents outlined could probably have been prevented had the men been wearing proper clothing and been provided with welder's masks. Similar accidents can be avoided by the use of an automatic bot of the type shown in Plate VI, *B*. Such a bot has been tried for two years at an increasingly large number of plants and has proved satisfactory even with excessively heavy limy slags, and with no danger of explosions when it strikes molten iron coming over the monkey.

At various plants cleats or ribbed plates alongside the cinder fall are provided to insure a firm footing. An accident several years ago shows the desirability of such cleats. A monkey boss slipped while botting the monkey and fell into the cinder fall or runner, which was full of cinder. He received severe burns.

Regarding accident 8, it is probable that an attempt was being made to change the monkey before the blast was off the furnace. in order to avoid delay. Many accidents have occurred in this way. It goes without saying that such a practice is to be discouraged, for although the change can be accomplished repeatedly with impunity, sooner or later the skull of chilled cinder, set up about the monkey, is sure to collapse and cause severe injury.

Probably no amount of foresight on the part of others than the injured men would prevent accidents like the last three mentioned. Injuries received in the course of handling tools and material are essentially due to some careless movement or method of work. Continual insistence on the need of personal skill and caution throughout the plant is practically the only method of preventing such accidents.

TAPPING HOLE.

1. Keeper was loading mud gun before cast, shoveling in clay while the helper operated valve. Clay plugged in the bottom of funnel and keeper put hand in to push clay down; plunger came back and cut off end of middle finger. C.

2. Helper filled clay cylinder of gun with water to clean the cylinder; turned on steam to push out the mud and clay; stepped to front and loosened stiff clay in nose with bar. Plug gave way, hot water scalding hand. D.

3. Helper turned steam into cylinder to clean it. The cylinder had been filled with water, and the hot mud shot out suddenly over a cinder snapper passing in front of gun. C.

4. Cinder snapper was in trough, ridding up after cast. Helper was taking gun from hole and turned on steam to help swing gun. Steam from exhaust burned cinder snapper's leg. C.



A. CINDER SHIELD IN ELEVATED POSITION. SHIELD IS KEPT UP WHEN CINDER IS RUNNING TO PREVENT ITS DESTRUCTION.



B. AUTOMATIC WATER-COOLED BOTTING MACHINE IN POSITION. OPERATOR NOT IN DANGER ZONE.

5. Helper was throwing clay into gun when steam hose extending from jib of gun crane to valve burst, scalding chest and arm. D.

6. First helper was swinging gun into hole. Nose of gun struck molten iron in hole and caused iron to explode. B.

7. First helper was swinging gun into hole. Nose of gun hit slag and iron lying about hole and caused slag to fly up, burning helper's neck and shoulder. D.

8. A cinder snapper stepped up to gun to turn on steam after gun had been clamped in hole, splash of cinder blew from hole and burned face and neck. D.

9. As keeper was operating valve on gun, cinder came back, flew, and burned wrist and hand. C.

10. Third helper standing between furnace column and clay gun was throwing clay into cylinder funnel of mud gun. As the load in the clay cylinder was discharged into the hole and the plunger was drawn back, cinder and gas came back and burned helper. B.

Other accidents occurring in connection with closing the tapping hole are discussed in another section.

Accidents to the feet or hands, like accident 1, occur both when funnels are used on the clay hole and when they are not. The only safe way of loading the gun appears to be to have only one man do the work. This rule is in effect at several plants, and has not proved impracticable, as the keeper, or other cast-house man assigned to the work, soon becomes adept at throwing and pulling the valve lever with his shovel. Although it would be possible for him to put his foot into the hole as he throws the lever, he is not likely to do so, and the method offers less chance of accident than any other except loading the gun from the nose, with a wooden plunger, which is more laborious, slower, and perhaps not as satisfactory.

As regards accident 2, the use of water to loosen and clean out stiff clay in the clay cylinder is common practice. Only the most careless foremen would permit a man to stand in front while the cylinder is being cleaned. The clay in the nose of the gun is always very hot and is usually stiff from being left in the tapping hole, with the lower part down inside the hot trough. It is better practice to pick out the hard clay in the nose with a bar before cleaning the clay cylinder.

As regards accident 3, aside from the carelessness of the helper, the management was deficient in not having the exhaust from the gun turned up and to one side, so that steam would exhaust away from men working in the rear of the gun, either during the cast when the steam cylinder was being warmed, at the end of the cast when the gun was swung to the hole by hand, or after the cast, when men were ridding up the trough before the clay in the hole had set and the gun was removed.

The fitting of the gun with pipe and swivel unions would minimize the probability of an accident like accident 5. The use of even wire-wound steam hose is not regarded with favor.

Since the introductory use of the Berg gun, in 1913, it has been adopted in many plants because of its efficiency and safety. The following extract, describing it, is taken from an article by Berg:*

The general method of plugging a blast-furnace tapping hole is to take the blast off the furnace, and stop the hole with a clay gun. The clay barrel of this gun is filled with clay before the furnace is tapped, and when all the iron and cinder has been tapped and the tapping hole begins to "blow," the blast is reduced to about 5 pounds pressure by slowing down the blowing engines; then the snort valve is opened, and sometimes the cold-blast valve on the stoves closed to take off all blast pressure in order to prevent gas from blowing out at the tapping hole, so as to allow workmen to insert the gun by hand and force clay from the gun into the hole by operating the three-way steam valve attached thereto, and by throwing small balls of clay into the gun until the hole is plugged, when the snort valve is closed and blowing engines are again signalled to go ahead at operating speed. This performance is not consistent with perfect safety of the workmen. * * *

The Berg gun is supported by a boom attached to a revolving sleeve on the furnace column and swings in and out of the hole by steam cylinders through cables. It swings horizontally by the first part of its travel, and then a chain attached to the rear of the gun tilts it during the latter part of the travel until the nose is on the same elevation as the iron notch; then the dogs on a shaft, turned by a steam cylinder, grab the gun and move it over the suspending trolley wheel into the iron notch and hold it there while the clay is inserted into the hole; the entire operation taking only one man, at a safe distance, to handle the different valves. * * *

The great advantage of this gun over the old method is, mainly:

1. The absolute safety of the workmen, as no men are exposed to flying sparks of iron or cinder, or the flames of burning gas as in the method generally used, or exposed to extreme heat while stopping the hole. * * *
2. Ability to stop the hole with full blast on whenever the furnace is working badly, with cinder or iron running back in the blowpipe. This is sometimes nearly impossible with the old method without exposing men to the danger of getting burned.

Mention should be made of the Gerwig automatic clay gun. This operates slightly differently, but is a prior invention.

The use of automatic clay guns is advantageous beyond question. There is, however, little occasion for injuries of the character noted in connection with stopping the hole, as methods in use at different plants can be combined to give safe practice.

Before a nonautomatic gun is swung into the tapping hole for stopping it after a cast, the nose of the gun should be warmed or coated with oil to prevent explosions when the nose of the gun comes in contact with any iron that may be in the tapping hole. Warming is usually done by pouring a little hot slag over the nose of the gun. The practice is sometimes criticised because if too much heat is applied, the heat is apt to stiffen the clay in the nose of the gun.

* Berg, H. A., The Berg automatic clay gun: Blast Furnace and Steel Plant, September, 1914, p. 231.

The danger from a plug of stiff clay is that the stiff clay may cause a "short" or "hard" hole, or if iron is laying in the hole, the iron may "come back" or run into the clay cylinder of the gun, and an "ironed hole" may be caused. Aside from the splash or spitting back of iron upon the men, there is a possibility of breakout at the tapping hole at the next cast when the hole is being opened, also the possibility that the hole will be so obstinate that the cast will be delayed far beyond the usual time for casting. The level of iron within the hearth will then become higher and higher, and unless the blast is taken off before the iron gets up to the level of the cinder notch, there is the probability that the monkey will be cut by the molten iron and will explode. Also, a hard or ironed-up hole during a cast causes increased danger of a hearth breakout as the iron gets higher. The downward and lateral pressure of the iron becomes much greater, and consequently the iron tends to penetrate the joints, interstices, and cleavage cracks of the brickwork.

When the end of the gun is covered with oil as a protective measure, any heavy oil is used and the nose is liberally daubed before the gun is placed in the hole. For some reason, even ice-cold iron covered with oil will not cause an explosion when it is plunged into molten iron.

Another excellent practice used at a few plants is to remove 1 to 3 inches of the clay at the front of the nose of the gun, and refill the cavity with sand or ground ganister in fairly stiff mixture with heavy oil or hot tar. The mixture eliminates still further the possibility of explosions of molten iron in the hole from wet or sloppy clay at the nose of the gun coming in contact with the hot iron, which may blow slag and gas back on the crew. The oiled or tarred sand acts as an inert barrier between any poorly mixed wet clay and iron with which it may come in contact.

INFREQUENT TAPPING-HOLE ACCIDENTS.

Other characteristic accidents, not of frequent occurrence, but of sufficient possibility to deserve mention, are as follows:

1. Splashes of cinder or iron caused by pushing the gun into the hole before the trough is drained.

When the cold or wet nose hits the molten material, an explosion or splash may occur. This may be avoided by giving the iron trough sufficient slope so that the slag and iron lay away from the hole when the iron and slag has been tapped and the blast taken off the furnace. It is usual to drain the trough before stopping the hole, but some few plants, to save time in stopping the hole or to get a better skim on the slag and iron remaining in the trough, postpone

draining until after the hole has been stopped. An adequate slope of the trough is then essential.

2. Explosions or splashes of iron when the nose of the gun is placed in the hole against a stream of iron.

Occasionally it is necessary to use the gun to help stop an unexpectedly large cast of iron. Explosion may be avoided by having the nose of the gun oiled, or warm and dry. To prevent splashes, a shield may be placed over the nose of the gun, ahead of the clamp bar (Pl. VII, A). The shield should be shaped to fit in the trough, being widened above the trough.

3. Flashes of gas and splashes of cinder when stopping clay does not fill tapping hole.

The shield mentioned above is an aid in preventing injuries from this cause. If shields are not used, a practice that should be followed is to form a fairly large lump of clay over the top and side of the gun nose. A simple method of avoiding "throw-backs" of slag, iron, and gas when the first load of the clay cylinder is discharged is to turn the steam on by a valve placed behind the column next to the tapping hole. A few plants use a long hook, thus enabling the keeper to stand off a considerable distance in shooting in the first load of clay. One superintendent insists that the keeper use this rod, not only for the first cylinder load of clay, but for the second reloading and discharging. The second helper, who has the duty of supplying the clay as the keeper works the plunger, should use a long-handled shovel to throw the clay in the funnel, instead of standing up under the blowpipe between the gun and furnace column.

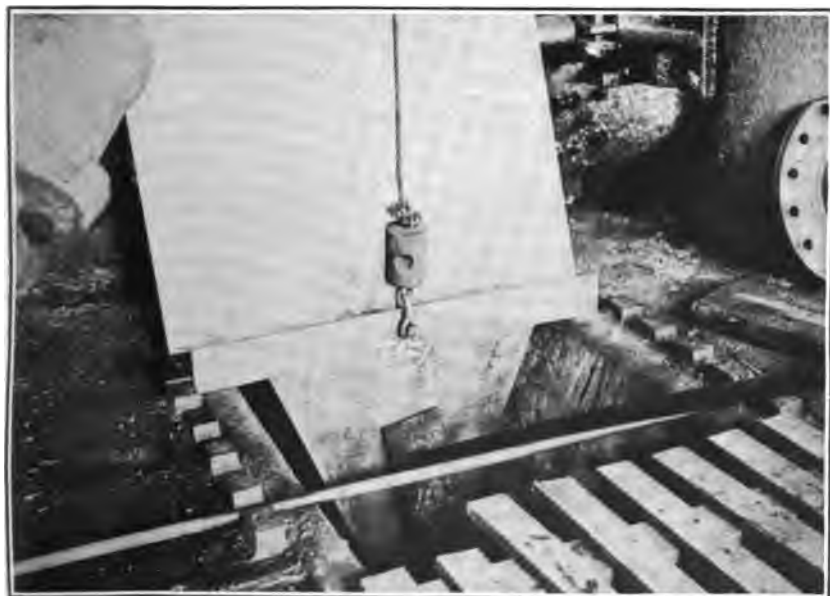
4. Injuries received on account of the hole blowing hard when the gun is put in.

Such injuries are extremely rare. With few exceptions, the opening of the snort valve, at the end of the cast, closes simultaneously a butterfly valve in the cold-blast main, just beyond the snort valve on the hot-blast stove side. Although the butterfly valve is not usually set to close tightly, on account of the apprehension that gas will come back in the hot-blast main from the furnace, the amount of air blown into the furnace is slight, being sufficient to keep up only an inconsiderable current in the stove and blast line.

Less in favor is a relief valve on the hot-blast line. This valve, when installed, is often clamped shut and never used. At a few plants, it is opened immediately after the snort valve is opened in order to stop instantly any blowing on the hole, thus obviating the need of putting the gun into the hole while the pressure is reducing through the tapping hole, or the need of delay until the flame dies down. If relief valves are used care is necessary in repairing them, as in one instance an explosion at the valve, with the gas drafted back



A. MUD GUN EQUIPPED WITH BELL-SHAPED NOZZLE AND STEEL SPLASH GUARD.



B. SHIELD FOR PROTECTION IN DRILLING TAPPING HOLE.

past it to a stove, knocked a millwright about 16 or 18 feet to the pavement. If the relief valve exhausts in the cast house, a ventilator with hood immediately over the valve should be provided to prevent the fumes from rising about the stack, where work may infrequently be necessary.

Whenever the flame at the tapping hole is strong and hot, it can usually be stopped by shutting the cold-blast valve of the stove on blast. The safest practice is to shut it to the position for filling the stove with wind, thus keeping some circulation of air in the blast-main system. It is never safe to shut down the engines entirely. Some costly explosions, both to life and property, have been caused by shutting engines down at cast time and leaving the blowpipes up and the tuyères open. A small quantity of wind is imperative, unless automatic check valves are used, or the stove is taken off the blast-main system. Even then valves may fail to work or some one may be negligent in his duties, when there is every chance for a blast-main explosion. Well-designed snort valves and cold-blast main butterfly valves do away with causes for injury at the tapping hole and prevent blast-main explosions.

ACCIDENTS CONNECTED WITH OPENING OF TAPPING HOLE.

Nine of the reported accidents involved in opening tapping holes were caused by the use of bars and sledges. A helper ran a sliver of steel into his hand while using a tapping bar; another's hand was hurt by splinters from a sledge; a keeper was struck in the ear by a sliver from a mushroomed bar. While holding a tapping bar one man was hit on the hand, two on the side, one on the forehead, and one on the temple. One keeper was hit on the wrist by a sledge flying off the handle. The disability ranged from 3 to 14 days, averaging 6 days.

Two men are usually employed to hold a tapping bar in order to keep the bar jumping and to prevent it from freezing in the skull, so that one man has to be on the same side as the man with the sledge. Broad-face sledges of the "Oregon" type are to be preferred for this work. Experience in sledging does not mean immunity from accident, for fatigue depletes the skill.

One plant employs tongs, the bar being placed in the hole and in the depression of a V bar, and held by men who, on opposite sides, grasp the bar with a long pair of tongs and who are enabled, with a little experience, to hold the end steady. In another plant the bar is placed on a crossbar over the trough, and is clamped down by bars held from opposite sides. The men, in addition to immunity from blows of misdirected sledges, are out of the way of any sudden outbreak of iron from the tapping hole.

Defective tools, such as mushroomed heads of bars and hammers, split sledge handles, loose sledge heads, and splinters of steel or wood, should be looked out for by the blower.

Twenty injuries were due to burns received while men were opening the tapping hole. Six of the injured men were experienced in opening the tapping hole with a hand or churn drill. From four to as many men as can get hold of the drill, which may be 24 feet long, are employed in opening tapping holes; usually six are employed. Three "spells" or turns at "drilling up" are commonly taken. The men next the tapping hole are bent over, and both their hands and their feet are exposed if the hole breaks through.

Two injuries, burns of wrist from hot clay blowing out and burns of ankles and feet from hot metal, were due to the iron coming out unexpectedly on the second "spell." The disability was 15 and 60 days.

Four injuries were caused by iron breaking or blowing out, unexpectedly or with unusual violence, in the third or last spell. The injuries resulted in disability of 6, 14, 30, and 30 days, caused by burns of fingers and wrist, arm and side, and left and right lower limbs. The experience of the men injured ranged from 3 months to 10 years.

Experience is of little avail to a man who takes a chance in opening the tapping hole, though it does contribute a certain intuition which guides experienced men in the handling of the drill as it approaches the skull of iron. Some oldtimers, however, will take chances, not only at the tapping hole, but everywhere about the furnace—chances that younger foremen will not assume themselves or permit others to assume except in most extreme necessity. However, the use of a hand drill up to the time the iron comes through is not very good practice. Not only are more men exposed to the possibility of burns, but a hand drill makes a bigger hole, which introduces a danger later on in the cast. It is much preferable to use a tapping bar to drive through the skull, after the crew have drilled out the clay in front of it. If a hand drill is used to get the iron out, the splasher should be placed, and a shield should be hung in the trough in front of the splasher. The shield can be slotted to admit the bar. A good practice is to lay a couple of sheets of corrugated iron over the trough up to the hearth jacket, with a hole punched in the sheet next the hole for the drill. The sheets should be laid over boards placed across the trough.

Four injuries were due to the use of air or electric drills. In one case an electric cable leading to a drill burned off as the result of a short circuit and the flash burned the keeper's hand, causing 18 days' lost time. Another keeper was burned by an outbreak in the tapping hole just as he was taking the drill from the hole. He lost 7 days

from burns of arm and wrist. The other two injuries were received while the men were still operating the drill motor, the drill biting through the skull and letting the iron out unexpectedly. The time lost was 10 and 11 days, caused by burns of chest and arms and of hand.

Some little experience is needed before an electric or air drill can be used with assurance of safety. Sometimes the drill will plug itself when it hits the skull. At other times it will bite through with no indication of approaching the molten interior. The safest practice is to use a shield in front of the tapping hole during the latter part of the drilling. The drill is withdrawn from time to time to remove the drilled clay from the hole. The use of compressed air to blow the dust out enables the use of a plate shield completely covering the trough and with a Z bar or angle riveted on at one end to be placed against the splasher. This device affords complete security. The front section with its angle may be made removable to afford a view of the tapping hole.

With a machine drill it is possible to make a much smaller and neater hole than is usually made with a hand drill. A 2-inch drill is standard for this work, although smaller ones are in use. By the use of a small hole the clay stopping in the iron notch is preserved against the attack of hot limy slag much better than when a large hole is driven down to the skull; the hole can be kept shorter, the use of an excessive amount of clay—with attendant uncertainty as to the length of the hole—can be avoided, and coke messes, flying coke and slag, and blowing out of the stopping during or after cast can be minimized.

Four injuries, causing loss of time of 6 to 10 days, were caused by the driving of tapping bars. No shields were used. In one instance a man was standing with one foot in the trough while he held the bar on which men were sledging. The skull broke in suddenly and iron blew out over his right foot and leg and also burned his hand. Lost time, 12 days.

In another instance the men had the tapping bar over a crossbar and were driving it in when the iron came out suddenly, hit on the crossbar, and splashed up on first helper's face and eyelids. Lost time, 7 days.

The third accident happened when a second helper was driving a tapping bar with a sledge. Flames shot from the tapping hole, burning his face and hands. Lost time, 10 days.

In the fourth accident the iron broke through quickly, throwing flame and hot clay over second helper's face, arm, and hand. Disability, 28 days.

The use of an adequate shield (Pl. VII, *B*) in front of the tapping hole should prevent such injuries in any plant where the bar is driven by hand.

Six injuries resulted from pulling bars from the tapping hole. As is evident from the review of accidents incurred while men were driving bars the iron at times comes out when the bar is driven. When the skull is heavy and thick and mainly composed of iron a "welshman" must generally be put on the bar, and it must be driven back by sledging on the wedge. Two of the injuries were to men having hold of the handle of the "welshman." In one case the hole was a little "green" or wet, and when the bar was pulled an explosion threw particles of hot iron over the man's face and neck. Lost time, 6 days. A similar accident in 1914 caused erysipelas, pneumonia, and death.

Two men suffered burns of shoulder, back, and arms from flames and iron from the tapping hole when the tapping bar was pulled. Disability, 3 and 24 days.

One man received slight burns of face and ear when the iron burst out while he was using oxygen to burn out a tapping hole in which a tapping bar was stuck.

The sixth accident occurred to a keeper pulling a tapping bar from the hole after the iron had come. The iron splashed up over his foot. Lost time, 16 days.

In pulling tapping bars the shield should be left in front of the hole until the bar is out; there is no necessity for removing the shield. In one plant, to enable the men backing the bar out to keep away from the trough, a sliding weight is placed on the bar and the men jar this against the "welshman" by yanking it up against the wedge with ropes attached to each side. In another plant, to enable men to keep away from the "welshman," a cable hooked into a clevis attached to the "welshman" by means of an eyebolt passes over a sheave to near the cast-house wall, where the "welshman" can be held at the desired height and pulled out of the trough when the iron comes and the tapping bar is free.

Men working about the tapping hole should always wear goggles and good shoes, tightly laced and not gaping away from the ankle or with cracks in uppers or soles. Leggings are used at few plants. They have, however, been found practicable in certain plants outside the State.

Men not actually employed in opening the hole should keep away, and if it is necessary to walk in front should get across the runner quickly. Some years ago a foreman was superintending work on a hard hole, and was standing directly over the dam in the trough. He was caught by a sudden outbreak in the hole, receiving burns

of abdomen and legs which caused several months' lay off. Care should be taken to keep the legs and feet out of the trough, even on the first spell, and while working to stand as much to one side as possible. It is safest to place a shield and splasher, or cover plate, in front of or above the tapping hole after the first spell, as it occasionally happens that the hole is weak, "short," or "green," and unexpected outbreaks of iron are almost inevitable from time to time.

TROUGH AND CINDER FALL.

1. Cast-house laborer was breaking a heavy skull of cinder in the runner when a piece of hot cinder flew, burning elbow. D.

2. Helper was crossing walk over cinder runner when an explosion of cinder in runner threw molten slag over back. C.

3. Cinder snapper stepped into cinder in runner while jumping across. C.

4. Water was dropping from bustle pipe onto cinder fall and when cinder was flushed it boiled, throwing material into keeper's face. D.

5. While man was tapping cinder, it splashed over runner into man's shoe. D.

6. Cinder snapper was throwing wet skulls of cinder into slag ladle when there was an explosion which threw slag into man's face. D.

7. Cinder snapper was attempting to turn on granulating-pit water-spray valve while hot cinder was running down trough. Heat from cinder set man's clothes on fire. B.

8. Monkey boss was cutting off cinder in runner from one ladle to another. and before he could get gate down tight his trousers caught fire. C.

9. Cinder snapper threw ball of clay into runner to plug leaky shutter. Cinder splashed, burning wrist. D.

10. Cinder snapper received burns of foot by explosion of cinder as he was throwing wet coal in runner to aid in running a stiff slag. B.

11. Second helper, in removing splasher, was pulling on hook to swing splasher around while others were on the lever. Splasher suddenly loosened and helper lost balance, falling into trough and burning body and legs. B.

12. First helper was placing splasher lever on splasher when hot metal was blown on him from tapping hole. C.

13. Helper was pricking tapping hole with bar when hot coke blew out suddenly, burning arm and body. C.

14. Helper was running pricking rod in tapping hole when obstruction in the hole suddenly gave way and he stepped into the trough which was full of iron. C.

15. Keeper was using rod on "mucky" hole when metal suddenly blew out, burning him. D.

16. Second helper was putting rod in tapping hole. When the cold or wet end of rod entered hole, explosion of hot metal occurred. C.

17. Keeper was cooling iron trough with hose so that he could rid up trough. Water hit pool of molten iron lying in trough, causing explosion. Eyes and face cut and burned. B.

18. First helper slipped into trough in which loam was boiling. D.

19. Keeper was loading clay gun. Plunger stuck, then flew back suddenly, causing steam cylinder head to fly off gun. B.

20. Cinder snapper pulled out lump of unchilled scrap from trough. Hot iron ran out on wet pavement and exploded. D.

21. Helper, in breaking crust on trough, let cold end of bar go through crust. It came in contact with hot metal in trough and explosion occurred. D.

22. Laborer, using bar to loosen scrap in trough, had torch hanging above him. Struck bar against torch, upset it, and spilled oil over clothing, which ignited. B.

23. Cinder snapper, in breaking cinder, slipped on rubbish and wrenched back. D.

24. Cinder snapper, while cleaning up about fall, fell and broke rib. B.

25. Helper let splasher lever slip out of hand onto foot. D.

27. Lever hanger broke and lever fell onto foot. C.

28. Cinder snapper let lump of cinder fall on toes. C.

29. Laborer, while shoveling cinder and coke from "mess," burned leg. B.

30. Helper, in sledging bar, hit keeper on knee. C.

31. Helper let bar fall on foot. D.

Accident 30 is a form repeatedly encountered all about the furnace front and runners. The use of broad-faced sledges for this work and standing on the opposite side of the bar from the man sledging it are about the only precautions to be taken in this work, but they will greatly help to eliminate this type of accident.

Keeping pavements cleaned up after each flush, cast, or mess is usual practice. Sometimes, however, accumulations of broken small cinder are allowed to collect. Keepers should keep floors and pavements clean.

The accidents in which men were injured while turning on water or operating shutters illustrate the type of accident in which men knowingly take a chance that involves their own safety when there is at stake the necessary performance of a task which, left undone, would probably result in expense to the employers for cleaning up the resulting mess or for repairing or replacing cinder-ladle equipment. Men are rarely asked or ordered to take these risks by their foremen: they take them voluntarily. Whenever safeguards can be installed to take care of contingencies that are fairly sure to happen, they should be installed. Spray valves for cinder pits should be placed at least 10 feet from a cinder runner and at least 20 feet from the pit itself. It is often possible for shutters to be so arranged that the cinder can be diverted by lifting the gate, not by having to jam it down, which requires a longer time. Better than hand-operated shutters are shutters that can be pulled or dropped by means of cables and sheaves at some spot away from excessive heat and out of the range of splashes or flying slag. When the shutter is dropped a guide is necessary. A steam-operated cylinder for cinder-runner shutters has for years been used at one furnace plant.

Boils in cinder runners are, fortunately, rare. When iron is coming over with the cinder, however, a damp spot in a runner will start a boil or explosion. To keep water from falling onto the cinder runner a shield, half circular, should be hung beneath the bustle pipe, di-

rectly over the cinder notch and extending out at least a foot on each side above the sides of the runner. This also prevents leaking or spattering discharges or feeds from running down over the bustle pipe.

The throwing of skulls of wet cinder into ladles of slag should be prohibited. It is just as dangerous to put them in empty ladles before running cinder, probably much more so, as they may cause an entire ladle of cinder to boil over or explode. Flipping or throwing coal or wet clay into cinder runners when the top is crusted is a frequent practice, though of doubtful advantage. The appearance of flame and occasional cracking and lifting of the crust as the steam or gas from the clay or coal is forced out is deceptive in that it seems to indicate a lively slag. Although the hazard of the practice is only trifling an occasional severe injury may be anticipated.

A steel-plate walk across a cinder runner at some place between the cinder notch and the ladle or pit is desirable. Men step across these places, and as the radiation and glare are frequently too intense to allow them to see and judge accurately their footing before they step, a misstep is occasionally made. The injury to the man crossing a footwalk just as an explosion occurred serves as a reminder that unnecessary approach to runners full of molten metal or slag is to be strictly avoided. Too much stress can not be laid on the requirement that blowers and crew acquaint themselves with the work so that there is a minimum of crossing and recrossing runners to get tools or to perform some forgotten job.

The best precaution against accident in removing the splasher is to pull it up and swing it away with a chain attached to a cable that passes over a sheave to a lever or steam cylinder at the cast-house wall. With this arrangement no one need be in a position to get burned or injured by a misstep or by the dropping of the lever. Another method is to suspend the splasher on a small trolley running up a monorail over the trough at an angle of about 60°. This device is, however, more expensive and offers the objection of suspending a heavy weight over men working about the trough between casts.

The relatively few accidents occurring from the use of the picking rod, which must be used at almost every cast, show that the hazard is not great. When a deep splasher is used and men are careful not to put their entire weight on the rod when it meets an obstruction, there should be little danger. Many blowers check the furnace when it is necessary to use the picking rod, and when there is, in their judgment, a chance of a burst of flame or hot slag, metal, or coke. The usual protection is a corrugated sheet placed upright alongside the trough, behind which the men crouch.

The closer this is to the splasher, and yet leave room in which to work the picking rod, the better the protection afforded. The picking-rod end should always be held over the slag or iron in the trough a moment to warm it before the rod is put into the hole. The safest way to bend the rod before use is to bend it about a column or between the gun crane and a column. A serious injury may be caused by one man standing on the rod while another lifts up on the end to put an angle on it.

Every year there are a number of similar accidents caused by putting water on the trough during ridding up. As the result of one on record a man lost the sight of both eyes. A man should wear goggles when turning water into the trough preparatory to cleaning it after a cast. The use of a special device on the hose adds to the safety. A 16 to 20 foot pipe is connected to the hose and on the end an elbow with a hose nipple or sprayer is placed. Every one should keep away while the water is turned into the trough. By moving the pipe, every part of the trough can be reached and thoroughly chilled. The work can then be finished up with an ordinary hose nozzle if desired. Another method is to place a holder on the end of the mud gun and to put the hose nozzle in it, turning on the water from the wall. However, this arrangement permits no flexibility.

RUNNERS.

1. Sampler put cold ladle into hot metal, causing explosion. Metal flew over keeper's leg. D.
2. Helper poured dipper of metal into sample box, causing explosion of hot metal which burned his abdomen. D.
3. As man was carrying cinder in ladle to dry runners, cinder slopped onto foot. B.
4. Keeper was running iron and when he lifted shutter an explosion or shot occurred, because the iron struck damp sand. Face burned. C.
5. Man was throwing shutter to divert iron into nest chill when iron splashed over face, neck, and arm. C.
6. Keeper had placed scrap in runner to melt it, and when metal struck scrap, explosion occurred, burning him. C.
7. Helper threw skull of cold scrap into runner of hot metal, causing explosion. C.
8. Helper threw runner scrap into hot-metal ladle about half full, causing explosion. D.
9. Helper was raising "punch-out" gate when iron exploded or shot, burning arms and shoulder. D.
10. Second helper was in line with tapping hole when a shower of red-hot coke blew from the tapping hole, setting man's clothing on fire. C.
11. Bricklayer accidentally stepped into runner full of hot metal. C.
12. Cinder snapper stepped on crusted slag and broke through, burning foot and ankle. D.
13. As keeper was watching iron run in ladle, spark flew into his eye. D.
14. Helper was struck in eye by spark from metal in runner. D.

15. Cinder snapper cut hand on scrap in lifting it into barrow. C.
16. Helper was loading scrap from runner into barrow and burned hand. D.
17. Cinder snapper picked up piece of hot scrap and had to drop it. Hit foot. D.
18. As two men were loading heavy runner scrap, one let his end fall, causing other man to drop his end of scrap onto his ankle. C.
19. Helper was carrying scrap when it broke in two and hit his foot. D.
20. Helper was loosening scrap in runner and struck hand against sharp scrap sticking up in runner. D.
21. Helper was claying up gate and cut hand on scrap sticking on side of runner.
22. Helper strained back in lifting heavy scrap from runner. D.
23. Helper was gathering scrap when he stumbled over a piece of scrap and cut his ankle. D.
24. Keeper punctured bottom of foot by sharp scrap cutting shoe. D.
25. Helper was breaking scrap with "mulligan," and when piece broke, one end flew and hit fingers. C.
26. Keeper was breaking scrap with sledge, which glanced off and hit his leg, splintering bone. B.
27. As cast-house laborer was sledging scrap, piece flew into his eye. D.
28. Laborer was sledging bar held by furnace helper. Missed bar with sledge and crushed helper's finger. C.
30. As laborer was holding bar to rid up runners, bar became wedged. Man put hand over top of bar to work it loose just as helper struck it with sledge. D.
31. Monkey boss had abscess form in palm of hand from handling bar. D.
32. Man was cleaning runners, prying hot scrap out with bar. Bar slipped, allowing man to fall forward into hot runner. C.
33. Helper threw bar down on cast-house floor, where it bounced and hit great toe. D.
34. Cinder snapper was barring scrap in runner, when he slipped and hit his shoulder on end of bar. D.
35. As man was barring heavy lump of scrap, bar slipped and fell onto instep. C.
36. Helper was lifting scrap with tongs, and when tongs slipped he fell backwards over wheelbarrow. D.

Accidents 16 to 36 were due to unskillfulness, forgetfulness, awkwardness, or misjudgment of the men, or to established but objectionable methods of work. The latter cause is subject to immediate correction. The former can be corrected only by sustained and continued effort to encourage care and prudence in this class of work.

Three men were overcome by heat while cleaning runners, and a fourth was overcome when running iron at cast. These accidents occurred in May, July, and September, months when periods of hot or sultry weather are encountered. As a rule, heat is not so difficult to withstand during a cast as during cleaning of the trough and runners, because with the sides of the cast house open and the almost universal ventilator in the cast-house roof, the heat is rapidly carried off when the iron is running, and, moreover, the heat is dry, in contrast to the moist heat frequently encountered about the runners. In addition, while a crew is running a cast, they only

infrequently have to approach or work close to the runner, whereas in cleaning the runners after a cast they have to work practically on top of the runners or troughs. These are always very hot and are frequently steamy, and when the cast has been a hot, limy iron, there is always a "heavy clean-up" on account of the excessive amount of scrap left in the runner. Owing to the length of the runner, fans are impracticable:

Most cast houses are all that can reasonably be desired in arrangements for admission of fresh air and escape of heat. Cast houses recently built are improved in that the fall or slope of the runners is much greater than in cast houses where iron was previously run in beds, also because the runners are much shorter from the skimmer to the first ladle junction. These changes lessen markedly the amount of scrap and the work necessary to remove it; hence the chance of injury or of heat exhaustion is also lessened.

An adequate and continuous supply of cool drinking water in the cast house is of assistance in reducing the liability of heat exhaustion or cramps. With water carried in pails, there is frequently excessive consumption of water every time a fresh bucket is brought in. If a drinking fountain is available, the difficulty of gulping down large quantities of water from the fountain, and the continual supply, tend to promote moderation in drinking, rather than the drinking of large quantities at infrequent intervals.

LIFTING SHUTTERS.

Five injuries occurred to men lifting shutters in running iron, owing to an explosion of hot metal when it encountered wet or damp sand, or possibly cold scrap when the gate was lifted and the sand worked up in the rear was pushed out or pushed aside by the iron. Disabilities of 2, 5, 18, 20, and 21 days were caused, owing to burns from flying metal.

The safest method of operating shutters is to run the iron into the ladle nearest the furnace first, and then into the next in turn, until the cast is finished. In this manner a shutter on the runner to the ladle is unnecessary, or is used only in a very fast and heavy cast, and the shutter at the junction in the main runner can be lifted by cables attached to the shutter and led over sheaves to the cast-house wall. The end is commonly placed one ladle away, on the ladle track side, down the runner. The keeper can in this position gage the fullness of the ladle above and operate the shutter at the correct time and from a safe distance. Some furnace men prefer to lead the cables to one point, near the furnace signal and lever, on the ladle track side. If a cast-house crane has been installed, overhead cables can not be used, but the cables may be led under the cast-house floor plates and attached to the shutter lever.

Devices to operate shutters from a distance can sometimes not be used if it is necessary or advisable, on account of possible need of shifting ladles, to run to the bottom ladle first. Under such conditions the gate must be dropped instead of lifted, and many times this must be done by hand, a mechanical device failing to seat the shutter against a fast run.

To avoid the necessity for anyone having to approach the junction and loosen up or bar out sand behind a shutter, it is necessary to put a foot piece on the shutter. This breaks a path for the iron by lifting the sand behind the shutter when it is lifted. To minimize the danger of flying sparks and splashes of molten iron in the event of damp sand, a shield should be placed on the lower side of the shutter to confine any spattering, as small explosions are not uncommon. If a shutter has to be dropped or thrown to divert the metal, care should be taken that it is dry. A hand shutter is usually thrown by laying it at the junction, with the gate over the runner, the handle extending back, so that the hot metal flowing beneath heats the gate up before use. Only experienced men should be allowed to throw shutters, or men who have had ample opportunity to observe how it is done, as there is a knack in gaging the throw, dropping the shutter, and turning away almost simultaneously to avoid injury from splashes. When the shutters are inserted by levers, it is essential to have the levers as long as practicable so that the men can keep away as far as possible. Care should be taken to remove all scrap from beneath the shutter, as when the shutter is dropped on heavy scrap and is forced down through it to make a tight stop, a boil may start.

Any dampness or cold scrap in a runner may start a "boil." Once started there is no telling where it will end. A boil in a sand runner has been known to eat down more than 10 feet into the fill, the boil being a continued succession of explosions, violent flames, and throwing about of hot metal and sand. In iron runners cold or wet scrap or damp loam or other lining material may start a boil that, for no apparent reason, continues to eat into the iron runner. A keeper dropped a gate on the runner and a sudden boil started. He received burns of the abdomen and was disabled for 19 days. Another cast-house employee, in running from a boil, tripped and fell, bruising his knee. He lost 4 days.

The keeper should always be sure that the trough and the runners are perfectly dry, and that no cold, heavy scrap is present before the cast is started. If loam is used to grout the runners particular care is necessary. A few plants use furnace gas to dry the runners. In using gas a good fire of kindling, coke, or waste should be started before the gas is turned on, and if the gas does not ignite at once,

it should be turned off and a hotter fire made. An accumulation of gas and air in the runners beneath corrugated iron will give a severe explosion if it is tardily lighted. As a second helper was attempting to light gas in the runner the gas exploded, burning his right ear, the side of his face, and his hand, causing him to lose nine days' time. Matches should not be used.

The practice represented in accidents 7 and 8 is hazardous in the extreme, but is sometimes indulged in when a particularly heavy or tough piece of scrap resists all efforts at breaking to size suitable for the stock house. It is better to send such pieces to the scrap drop, and blowers should forbid the putting of scrap in runners either before or during a cast unless they are present to see that the scrap is dry and warm. Putting scrap in ladles should be avoided absolutely, for, aside from the hazard, it accelerates the formation of a skull on the ladle.

Such "shots" or explosions as are illustrated in accident 9 are due to moisture and occur in the same manner as when runner shutters are operated. The best safeguard is to lift the gate by means of a lever or steam cylinder at the wall. At one plant the same cylinder is used to remove both the splasher and the punch-out gate. When this arrangement is not feasible, on account of an overhead crane, other methods to insure safety may be used.

One method is to notch the bottom of the gate, and at the end of a cast, after the removal of sand banked against the gate on the outside, the iron is drained from the trough by using a long picking rod to poke out the clay behind the notch at the bottom of the gate, the man thus being away from the trough on the cinder-runner side, in contrast to the usual position in which the helper pulls the gate up a trifle; and then, holding the lever by one hand, punches a bar through the clay or sand at the bottom to drain the iron. In the latter case the bar must be pushed through the molten iron in the trough, and the helper, on account of the angle at which the bar must be held, is close to the gate in a place where any shot may easily catch him. When this method of raising the punch-out gate and draining the trough is employed an effective guard may be used. This consists of a cast-iron plate held at an angle of about 45° by a couple of supporting plates cast on the sides. A hole is left in the center of the plate for the punch-out rod.

Another method, little in use, is to cast the punch-out gate with a round hole at the bottom. This is heavily grouted and plugged with an iron "bot." The iron is drained through this hole until cinder comes, when the hole is botted up and the gate lifted. The objection is that the gate is too frequently cut out by the iron. This objection is in itself unimportant, as far as expense of renewal goes, but there is the possibility that during a cast a leak of iron may start through,

become unmanageable, and drain more or less of the entire cast onto the first ladle and onto the yard. This development is much dreaded with any kind of a punch-out gate, and in some plants a double gate is used, seemingly with satisfaction. It goes without saying that both the inside and outside of such gates are very carefully clayed and banked up.

One advantage of this method of draining the trough is that it makes the skimming much easier at the monkey skimmer. This is usually a hot job, and there is frequently a small explosion when the sand dam is knocked out to drain the slag away and to keep it from going into the iron ladle. If the iron can be largely drained from the trough before the overlying slag comes, the separation of the slag and the iron is much cleaner and less difficult; hence the liability of accident is lessened. Probably the safest method of accomplishing this secondary skimming at the end of a cast is to replace the old-fashioned sand dam with a cast-iron dam, which can be pulled up by a lever or with a coned-out shutter, which serves the same purpose. No injuries are reported specifically as occurring at the monkey skimmer, but it is probable that at least one of eight unclassified burns received in the cast house occurred at the monkey skimmer.

Eight burns received "while casting" resulted in lost time of 4, 4, 5, 9, 15, 21, 39, and 40 days, the injuries being burns of (1) instep, (2) back, (3) eyelid, (4) foot, (5) eyeball and eyelid, (6) eye, (7) heel, and (8) foot, back, and ankle. Lack of safeguards or unsafe practice may have caused them, but five, at least, might seemingly have been avoided had the men been wearing well-adjusted goggles and tightly fitting sound shoes.

Many of the following points have already been brought out in the discussion of "accidents connected with opening of tapping hole."

It is most difficult so to arrange a cast house that all necessity for men's crossing in front of the tapping hole is avoided. A man should cross quickly and should avoid standing in the range of pieces of coke, hot slag, or metal that might fly from the hole if it becomes too large or starts to blow suddenly. The danger of material flying from the hole can be minimized by using two splasher plates. Their use is not as common as it should be. A further safeguard is the provision of a substantial steel-plate shield over the skimmer, or, if preferred, further down the runner, to protect the men running the iron. Such a shield is especially desirable if the shutters are in line with the tapping hole, and is well warranted in any cast house on account of the necessity of occasional passing in front of the tapping hole. The only objection to placing the shield at the skimmer is that it may deflect a burst of flame or coke on men working alongside the trough with a picking rod to open a hole.

Care should be taken to keep off from crusted slag.

Accident 1 is the only specific mention of injury caused by the use of cold bars or ladles in running hot metal. It is probable that others of the unclassified burns were from this cause. The danger of the practice is well known but new men should be informed of it.

Goggles should be worn by cast-house men while running iron and while breaking scrap.

OTHER CAST-HOUSE ACCIDENTS.

1. Machinist had hand on mud-gun clamp shaft which was being lifted with crane. Clamp on shaft slid and caught fingers. C.
2. Cast-house laborer was fastening cable about scrap; signaled craneman to hoist but kept hand on cable and was caught. D.
3. Helper was unloading clay from car with crane bucket and was caught between side of car and bucket when craneman hoisted. C.
4. Foreman was laying iron runners in cast house. Sling rope on tackle broke, allowing runner to swing about and catch man's leg, amputating it. B.
5. Machinist was holding incandescent light on extension cord, when socket short-circuited, burning his hand. D.
6. Roof cleaner on cast-house roof fell through to cast-house floor. A.
7. Laborer was helping take down chain block used for setting runners, and fell to cast-house floor. C.
8. Blower was hit in eye by spall from mushroomed head of wedge on mud-gun clamp. B.
9. Laborer stepped into loam box full of boiling loam. C.
10. Helper was scalded by steam from burning hose on hot scrap. D.
11. First helper's hand became sore and infected from using bar. D.
12. Cast-house laborer cut hand on scrap and did not report for two days. Infection. D.
13. Helper tripped and fell into hot sand of iron beds. D.
14. Pig-iron carrier, in polling iron along runner, made misstep and fell on side of car. B.
15. Pig-iron carrier dropped iron on feet. B.
16. As pig-iron carrier was breaking iron, pig caught on apron, throwing him onto breaking block below cast-house sill. B.
17. Pig-iron carrier was walking up plank to throw iron into car when plank rocked, causing him to crush hand between iron and cast-house door. D.
18. Laborer stepped into hot sand. C.
19. Laborer got hand full of hot sand in carrying iron. D.
20. Helper was running iron in beds, when explosion occurred, throwing hot metal into his glove. D.
21. As man was loading scrap into wheelbarrow, a glove caught on scrap. D.
22. Man carrying scrap allowed it to fall onto his feet when he stumbled. D.
23. As man was wheeling scrap on wheelbarrow, piece slipped off onto foot. D.
24. As man was throwing scrap into box, he cut his knuckle and infection resulted.
25. Laborer, in moving clay gun, caught his finger between blocking. D.
26. Brick pile fell over onto man while he was removing brick. D.
27. Man caught foot between casting and runner. B.
28. Men were lifting floor plate onto truck and allowed it to fall onto laborer's foot. D.

29. Helper picked up hot bar and burned hand. D.
30. Man stepped on bar and twisted his ankle. D.
31. Laborer was swinging sledge to break scrap and strained muscles of chest. D.
32. Laborer was wheeling wood in barrow, and piece of wood with nail in it slipped off and man stepped on it. D.

Mention of accidents of the type of 10 to 30 might be prolonged to give many variations. As regards development of safeguards, such a review serves little purpose, for safeguards are not possible. As a rule the injuries are not serious, and in a way, such accidents might be considered a hazard of the work. The possible usefulness of a review lies in the emphasis it places on the necessity of carefulness on the part of the workmen. The majority of accidents result from handling scrap, followed in order of frequency by dropping material while handling it, and by stumbling. There can be traced no relation between the length of employment and the liability to injury from such an occupation as hand labor and the use of hand tools. Only in injuries incurred in the handling of hot bars and scrap does the term of employment appear to have any bearing. Five of six men injured had been employed less than six months.

Precautionary measures suggested by the preceding review of accidents include a frequent inspection of equipment, particularly rigging tackle, cables, ropes, chains, and blocks, to see that they are safe. The supervision of rigging used can not be too painstaking, for life and limb will often depend on its strength. Each incandescent-light bulb on an extension cord should be provided with an insulated handle, away from the lamp socket, with which to grasp the light. A wire guard about the bulb should also be provided.

Men working on roofs should wear belts with safety lines. If it is not practicable to secure the end by tying it to a roof ventilator angle, some one should be delegated to hold the loose end. Especial care is necessary in walking over corrugated-iron roofs of cast houses after they have been in service for six months or more, because the steam and sulphur fumes from the furnace casts and flushes quickly rust the iron and, although it may appear substantial, it may be a mere film of iron crusted with rust. Plate roofs are preferable and are used in modern cast houses, where they are given a slope sufficient to obviate the necessity of cleaning.

Falls in the cast house over bars, scrap, and cinder can largely be avoided by keeping the cast house clean. Racks should be provided for all tools. Burned bars should be removed promptly or set along the wall after each cast, and sledges should be put away from places of work or travel. Constant effort should be made to prevent accumulation of rubbish and tools underfoot. Neatness carries with it a certain self-discipline and, whereas a dirty, cluttered-up

cast-house floor breeds carelessness and increases the liability to accident, a constant cleaning of the floor helps make men careful in other ways.

Many of the injuries resulted from methods of doing work that were relatively inefficient as regards the least effort, quickest dispatch, and maximum accomplishment. Training men to make improvement on the economic side has shown benefit in that the men take precautions against accident as their skill and alertness of mind increases. Such training sets men's minds to work, and when that has been done the men comprising a cast-house crew, or other gang, begin to take an intelligent interest in real preventive measures against accidents.

DISPOSAL OF SLAG, FLUE DUST, AND IRON.

FLUE-DUST DISPOSAL.

Flue-dust plants, at which the dust from dust catchers and from the flue-dust stock pile is sintered, nodulized, or briquetted, are becoming standard equipment at many furnaces. They vary from simple installations to those involving considerable intricate handling machinery. The handling of flue dust causes many burns, as it may be at a temperature of 300° to 500° F.

1. Helper fell down steps, spraining wrist. D.
2. Attendant tripped over loose board, bruising arm. D.
3. Otter removed cover over chute and failed to replace it; later stepped through it. D.
4. Laborer was on ladder when it slipped. D.
5. Foreman was leaning against railing when it gave way, and he fell to track. B.
6. Laborer stumbled and fell against hot plate. D.
7. Laborer in handling brick had abscess form on hand. D.
8. Laborer's hand became infected from handling wheelbarrow. D.
9. Inspector was holding wedge for fellow workman to strike with sledge when head of sledge flew off handle, striking inspector on head. D.
10. Millwright's helper was removing cupboard, when it fell over on leg. C.
11. Laborer was putting belt on pulley in motion and caught finger. D.
12. Conveyor man attempted to remove dirt from tail pulley while conveyor was in motion; hand was caught and arm broken. B.
13. Pressman in loading briquets on buggy put foot on rail; operator started buggy, crushing and breaking pressman's toes. B.
14. Laborer in unloading flue dust burned back of hand. D.
15. Laborer in unloading flue dust slipped and fell into pile of hot dust, burning arm, neck, hands, and leg. C.
16. Laborer stepped into pile of dust along track and got hot dust in shoe. C.
17. Unloader jumped off top sill of car into hot dust. B.
18. Unloader opened drop doors of car of dust, and hot dust flew out, burning hands, neck, and face. B.
19. Laborer was pulling nodules down chute into car, when rush of material caused hot material to fly into face. C.

Beyond equipping steps with handrails and nonslip treads, it is difficult to eliminate the probability of slips and missteps. Statistics over a wide range of industries indicate that of all accidents falls down steps comprise approximately 4 per cent, and falls from ladders approximately 4 per cent, and that approximately 40 per cent of all falls are due to slipping and stumbling on the ground level. Thorough inspection of railings at regular intervals, keeping yards and floors clean, eliminating depressions and projections, and replacing or guarding manhole covers and pits is important.

Accident 9 illustrates negligence of employer in that tools should be regularly inspected for defects. Other hand-labor accidents were largely the result of lack of skill or of carelessness. The same is true of those accidents in which men were injured by machinery.

Hot flue dust will fly into the air and run on the ground to a greater extent than other material usually handled about furnace plants. It is deceptive in that it usually does not appear to be hot, and it seems to possess the property of clinging to the skin when it comes in contact with it in flying through the air, accounting for the seriousness of flue-dust burns. Men engaged in unloading hot dust should wear tight shoes with trousers fastened about the ankles. The sleeves of coats or should come down to the wrists and be fastened over the tops of the gloves. Fastenings for trouser and sleeve bottoms should be easily and quickly removable, rubber bands are satisfactory.

Men should not be allowed to jump into cars of flue dust to shovel it out, for although the top dust may be cold or damp, the interior, below a thin crust of cold dust, is often exceedingly hot.

Some adequate spraying device at dust catchers is urgent. A simple ring spray is apt to cause excessive saturation, so that the dust is unfit for use at the flue-dust plant. Consequently the dust is sometimes much undercooled, or is cooled only in sections. Spraying the dust in the car after the dust bell has been dumped is not satisfactory because the top is saturated, but the bottom and interior may remain hot. A device in use is an inverted truncated cone inclosing the spray and bell. This accomplishes the work more efficiently than any other device designed to dampen the dust only partly rather than to completely saturate it.

Walks along and over flue-dust receiving bins are preferably made of cast-iron grids, extending from the rail girder to the sides of the bins. These grids prevent men from inadvertently stepping into heaps of hot dust. One concern has a movable plate shield which is placed alongside the car-door hopper to prevent dust from being thrown out sideways in bulk when the doors are opened.

cinder cools rapidly, and after 24 hours the entire contents may be safely assumed to be solid. One company requires that stickers be sidetracked at the dump for 72 hours before they can be worked on. Such a requirement tends to make the ladle gang put forth special effort to keep ladles in good condition, so that there will be the least chance of stickers, and is a requirement that keeps dangerously cracked or warped ladles off the run.

After the liquid contents of a tilted ladle have drained an occasional practice is for the cinder-dump gang to step to the front and bar out the skin skull. Usually only a few prying with a bar are required to start this skull, and the men develop skill and agility in performing the work. However, occasional burns are thus caused, and it is better practice to jar the skull out by tilting the ladle back and forth against the frame. This procedure is, of course, practicable only with steam-dumped thimbles. The skull of thimbles dumped by hand or pushed by a locomotive should be allowed to cool, as the contraction and cracking of the cinder will often cause the skull to slide out, or lessen the chance of burns if the skull has to be pried out by hand. In 1914 a man was severely burned when he stumbled as he started to jump back out of the way of a hot skin skull that suddenly collapsed.

An occasional cause of accident is an explosion of cinder ladles. As a rule the explosion occurs when the ladle is standing at the cast house and during a cast or flush. It is caused by damp rubbish or water being in the bottom of the ladle before cinder is run into it. Usually nothing more serious than a violent boiling and splashing takes place, which, as a rule, starts slowly. When iron goes into the ladle with the cinder, however, a veritable explosion may throw the entire contents of the ladle over the yard and into the cast house. In 1914 one man received minor burns by such an explosion. Such explosions can be prevented only by carefully examining the ladles when they are spotted at the furnace, and, if they are wet or contain water, by drying them with hot cinder or a wood or coke fire. Damp rubbish should not be thrown into ladles.

Similar explosions are sometimes encountered in cinder-granulating pits, two men being seriously injured in this manner in 1913. They are caused by iron running into the pit in large amounts. Men should keep away from the pit as much as possible during a cast, and should never go near it when the skimmer is being drained at the end of a cast.

An aid to stop stray iron from reaching ladles or granulating pits is a catch basin between the main and the monkey skimmer, and the ladle or pit, which will catch iron going over the dam or iron that has been poorly skimmed. Such basins prove their efficiency by

the considerable amount of iron recovered in them, ranging from 5 to 40 tons per month per furnace.

Such accidents as No. 20 are liable to occur if the crane cab of cranes is placed over the pit. As a rule, the span of the crane girders is sufficient to allow the cab to be placed on one side of the pit. If this placing is not possible, as is true with some types of cranes, the craneman should not be allowed to run over the pit to dig up cinder while the furnace is flushing or casting into it.

Considerable necessity for work about cars of granulated cinder may be obviated by having a deflecting plate between the pit and the cars of dust. So far as possible, track cleaners should wait until the filled cars have been removed by the engine before cleaning tracks. The danger of hot water from cars and from buckets inadvertently carried overhead is thereby eliminated. There should be no pathways beneath the swing or travel of buckets handling granulated cinder.

IRON DOCK.

The iron dock, the iron wharf, and the pig-storage yard are terms designating some part of the furnace yard or the breaking platform at the base of the cast-house wall, or in the yard where the iron is loaded or stored pending shipment. The injuries result largely from handling the iron pigs by hand, though the introduction of locomotive steam cranes and lifting magnets has introduced a hazard that may cause injuries more serious than are typical of handling by hand. However, such injuries are of much less frequency.

CRANES.

1. Laborer, loading pig iron with crane and standing at end of car with hand on car door, signaled crane to move up. Crane struck car and caused door to pinch finger. D.

2. Fireman was standing with hand on brake wheel of car being loaded with pig iron, when crane operator accidentally dropped magnet on man's hand. Left hand contused and lacerated. D.

3. Laborer was collecting scattered pigs in car for magnet to pick up, and as magnet was lowered into car man attempted to pass between magnet and side of car, when he was caught by swing of magnet and crushed. Fractured hip. B.

4. Crab man unloading iron from car into yard with magnet did not see magnet swing back after it had discharged load of iron and was struck by magnet. Compound fracture of wrist. B.

Locomotive-crane engineers working in cabs inclosed in the sides and with their minds fixed on the manipulation of hand and foot levers can not give entire attention to possible hazardous positions of men working with the crane. The movements of the crane and the magnet are fairly regular and may be anticipated. Men, working

with the crane should time their work to the craneman's operation, and signal for a stop if necessary. Foremen or shift bosses should impress upon the men the necessity of keeping from under the travel of the magnet, of going to the other end of the car when the magnet is lowered and raised, and of having the crane stopped when it is necessary to collect the iron.

FALLS OF PERSONS.

1. Iron piler, 10 years' experience, stumbled and fell on hand. Dislocated finger. D.
2. Iron piler, 8 years' experience, stepped on wet spall and fell against a pile of iron, cutting arm. D.
3. Iron loader, 6 years' experience, stepped from top of pile against iron used to hold plank in side of car. Deep laceration of thigh. D.

FALLING AND FLYING MATERIAL.

1. Laborer, 4 years' experience, threw pig from car on pile, and spall broke off and flew in eye. D.
2. Laborer, 2 years' experience, threw pig on pile, and sliver flew and cut eye. D.
3. As laborer with 11 years' experience was handling machine cast iron lime dust from pig flew into eye. Burned left eye. D.

HAND LABOR.

Eleven injuries to the hand and fingers were caused by hand labor. Five were caused by the men getting their fingers caught by pigs rolling or falling down a pile. The disabilities were 3, 7, 11, 19, and 20 days, caused by cuts of finger, cuts of palm of hand, loss of nail, crushed and cut finger, and crushed hand. The experience of the men injured varied from 5, 10, 2½, 10, and 14 years.

Two injuries were due to piling or loading pig onto hand, resulting in crushed middle fingers. The time lost was 7 and 14 days, and the experience was "years" and 3 hours.

Three injuries were caused by striking the hand against the top or edge of a car or against another pig while carrying iron. Loss of time of 7, 7, and 15 days, caused by cuts at base of thumb and on knuckles. One man of 4 days' experience cut his finger on a sharp edge of pig iron. Lost time, 5 days.

Nine injuries resulted when men let pigs drop on their feet when pigs slipped from hand leathers or caught on hand leathers, or a man stumbled or bumped his body or arm against some object.

Five injuries were caused by pigs rolling down from piles onto men's feet.

One laborer who had had 10 days' experience suffered a strain while lifting iron to load it in a car. Lost time, 13 days.

As regards such hand labor as carrying iron, it is of interest to note how prone men are to accident even after they have had considerable experience.

PIG MACHINE, LADLE HOUSE, AND SKULL DROP.

PIG MACHINE.

1. Lime man was using winch when cable tended to run off grove. In guiding cable with hand, fingers were caught between cable and drum. D.
2. Ladle man placed crane hook on lug of runner and gave hoisting signal. When hoist was made, his fingers were caught between chain and runner, and his little finger was fractured. B.
3. In tilting ladle to pour, pig-machine man kept fingers between hook and lug of ladle. Thumb and fingers were lacerated. D.
4. Laborer, in wheeling scrap in cellar of machine, slipped on wet pavement and cut lip on scrap. D.
5. Laborer in adjusting pigs that projected over side of car slipped and fell from top sill to track, spraining ankle. B!
6. Troughman, in cleaning runner, had foot on bar to pry scrap. When scrap loosened, bar fell on other foot. D.
7. Scrapper was punching sticker out of molds. Pig fell out and broke and flying fragments hit man's eye. D.
8. Laborer passed under mold chain while machine was in operation. Wheel fell on head. D.
9. Carman was standing opposite chute taking car numbers while cars were being loaded. Spall flew from a pig dropping from chain to chute, and cut eye. C.
10. Pig-machine laborer was holding chisel to cut cotter pin from wheel pin or shaft. End of pin flew and cut eyeball, causing loss of sight. B.
11. Troughman got hot graphite in eye while casting. D.
12. Helper was carrying scrap from pouring end, and tore thumb. D.
13. Pourer was throwing scrap from conveyor tough, when foot slipped and he fell with the scrap. C.
14. Machine laborer was holding bar to cut sticker from mold. Helper missed bar and struck man's nose with sledge. D.
15. Ladleman was operating dumper lever and struck hand against nail protruding from controller box adjacent to lever. Infected puncture of hand. D.
16. Eye of laborer unloading lime became inflamed from lime dust lodging on the eyeball. D.
17. Hot spray of milk of lime was blown into foreman's eye from lime vats. C.
18. Laborer accidentally stepped into tank of hot limewater. D.
19. Lime man was turning off steam line at vat when foot slipped, causing him to plunge arm into vat of hot lime water. C.
20. Lime tender was stirring lime in lime box with hoe and slipped into vat. B.
21. Pig sticker became enveloped in steam cloud and stepped into hole of hot water. D.
22. Carman, in descending from casting end by ladder, was caught in projecting set screw on rear sprocket wheel drive shaft. B.
23. As helper was barring scrap from mold-chain rail beneath spout hot metal splashed on him. C.

24. Hand became infected beneath callous. D.
25. Iron ran into mold containing cold sticker, causing explosion that blew "sticker" out of mold into operator's shanty, hitting him on head. D.
26. Pourer was struck on hand by hot metal flying from explosion of hot metal in wet trough. D.
27. Man was pouring pig-machine wheels when mold exploded, burning back. D.
28. Pig-machine helper was burned by explosion of hot metal when it ran into cold or wet mold. D.
29. Man was cleaning up in cellar while ladle was being poured. Considerable iron ran over mold; hit water on floor of cellar and exploded. D.
30. Helper was polling iron in trough and a splash or spitting of iron burned eye. D.
31. Helper's goggles became clouded with steam. He pushed them back on forehead when a small splash of metal hit eyes. C.

Accidents similar to Nos. 1, 2, and 3 are common in iron and steel plants. Perhaps the primary cause is that the man, after making a hitch about an object, fails to keep a certain tautness in a chain or cable until the crane tightens up on the line. Care should be taken in doing this to keep the hand away from any side obstruction, and especially to keep it above the hitch so that it will not be caught or struck. When simple hooking on is being done, a safety hand grasp should be provided on the back of the hook, as injury appears to be ultimately almost inevitable when men have to grasp the body of the hook in guiding it.

In working about places that are apt to be littered with scrap and to be wet from spray water, care should be taken not to overload the barrow and to clear a path for the barrow when wheeling out scrap. No one should be allowed to walk on top of the sides of cars, as the sills are apt to be warped and crooked, causing a man to lose his balance. Barring down scrap tightly stuck to runners is, at best, somewhat venturesome. It is work that has to be done, nevertheless, and in spite of care a bar will occasionally fall when the scrap comes loose or gives way suddenly. New men should be warned and instructed in the safest way to do the work. Jumping on or pushing down bars with the feet is the most dangerous way.

Practically the only safeguard to prevent such accidents as No. 9 is a screen alongside the car track opposite the pig-machine chutes. Screens at the end of the conveyor or strands, where the pigs drop onto the car chute, are equally efficient, though they do not prevent spalls from flying out of the car when the pigs drop on top of the iron already in the car. However, they ~~protect men~~ working at the end of the chutes punching out stickers, where mechanical ~~knockers~~ are inadequate or unprovided. One man was severely injured in the eye by a flying piece from a pig that hit a manganese-steel chute.

Another safeguard that should be standard is a plate canopy beneath the mold chain, between two adjacent structural columns.

Men can pass beneath the canopy or can stand under it while watching for stickers when the machine is in operation without any danger of pigs dropping on their heads. (See Pl. VIII, A.) A rail should be placed along the rest of the side columns to keep men from going under the chain at any point except where the canopy is placed. The plate can not extend indefinitely from front to back, and to save a few seconds some men will take a short cut unless prevented.

The somewhat costly safeguards mentioned prevent only a small proportion of injuries, some of which, possibly, have not yet been encountered at many plants. The safeguards should be considered as a part of the general investment that is calculated to reduce insurance rates, not as isolated devices of problematical usefulness. A further precaution should be the requirement that men working beneath a pig machine or at the casting end, or engaged in sledging or cutting, shall wear goggles.

A safety committee of workmen, consisting of an electrical millwright or handy man, a pig-machine pourer, a cast-house helper, and a trestle, yard, shift, or "straw" boss, if they are thoroughly interested and have authority to correct dangerous practices, and are familiar with the character of such accidents, can do much to prevent accidents, not only at the pig-machine but all over the plant, as at the trestle, the stock house and the cast house. Haphazard, unstandardized methods of work, rather than carelessness, are often the essential cause of these accidents. The foreman, even more than a committee, must correct such methods of work; must show the men how to carry scrap efficiently, how to stand and swing a sledge effectively, and how to hold a bar to best advantage. Hand-labor and hand-tool accidents are said to be the most difficult to foresee and prevent. Perhaps more can be accomplished by training the force to a high degree of skill in their routine, every day work than by specializing in a purely safety campaign, as usually a capable man is a careful man.

Accident 31 emphasizes a general complaint in regard to goggles—that they become steamed and must be removed and wiped. Often the necessity of attention to the work does not permit immediate cleaning and they are cast aside. Many companies say that they have been unable to obtain goggles without this defect. If the steaming of goggles becomes insufferable, butt welders' masks should be used because the considerable number of eye injuries indicates that some eye protection is essential. If goggles are slipped over the eyes only when the wearer approaches or faces hot metal, the effects of steaming are not so noticeable.

Attention should be given the footwear of men working about casting operations. Tightly fitting shoes, without cracks, should be

worn, and trousers of wool or hard jean should come down over the tops. Leggings are feasible, but are not looked upon as necessary by the great majority of companies.

Explosions of hot metal poured into cold or wet molds are common. When the casting machine is being used regularly for each cast, there is the least danger of such explosions; but when it is used only infrequently for excess or off-grade iron, there are often noisy detonations and considerable flying of sparks, and occasionally a violent explosion occurs. Wet molds can be largely prevented if the roof over the casting machine is kept free from leaks, and in very cold weather excessively cold molds can be avoided by a coke jack or brazier under the mold chain. Flying sparks from wet or cold molds can be held down by shields over the runner spout where the iron drops into the molds, the shields being similar to those mentioned for cast-house spout junctions and punch-out gates.

The pourer's window is best protected by a heavy screen. Trough men can only infrequently work from behind shields, though some plants have provided corrugated-iron shields having a hole through which the trough man can ravel or pole the graphite in the runners. To permit trough men to get from one side to another of a double-spout pouring trough, a crossover above the chains should be provided. Care should be taken to see that the runners are dry, and before the iron is poured any crust of cinder or coke dust on top of the iron in the ladle should be broken up, so that it will not hold the iron back and form a dam that will break through and allow the iron to pour out faster and with a greater splash than is safe. Only long poles should be used, and men should avoid as much as possible getting close to the runners, especially when the ladle is about to be tilted.

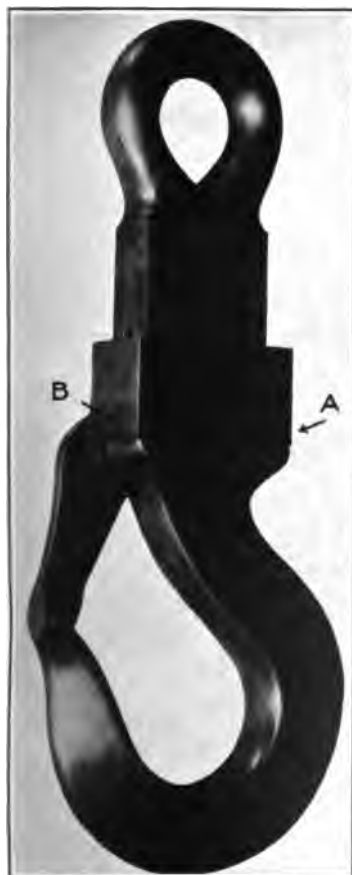
Men unloading lime should be provided with goggles with solid side pieces instead of gauze, and with no gaps at the top, sides, or bottom. A lime burn, either of the eye or skin, is apt to be severe and of long duration. Valves for lime vats and lime sprays should be so placed that they can be operated from beneath the mold chains at a safe distance from the mixing box. It is preferable to mix the lime in a box adjacent to the lime storage bin, whence it may be run to the vats beneath the stands. This arrangement tends to promote safety and efficiency in getting a rich milk of lime with no foreign matter in it to cause clogging of the sprays, as frequently occurs when the lime is mixed in the spray vats. The mixing box should be protected with a railing and foot board. For convenience, the rail should be hinged so that it can be turned up when the vat is to be cleaned.

Holes and sewers, especially those carrying hot water, should be provided with gratings.



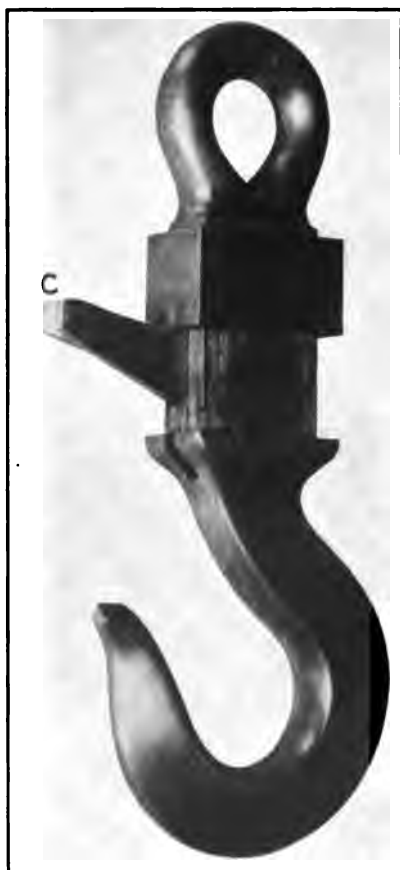
A. SAFEGUARDS FOR PIG-CASTING PLANT.

A, Signs posted near track; B, covered passageway beneath molds; C, fence to prevent men from taking other than the covered passageway.



B. SAFETY HOOK FOR HOISTING BUCKETS.

Sleeve A fits into notch B and locks hook in closed position.



C. HOOK OPEN.

Latch C drops and is locked automatically.



Recent pig machines have steps provided for access to the rear platform. Projecting set screws should be replaced with counter-sunk screws or have a guard placed about them.

Although many plants do not place rails immediately beneath the spout, thus doing away with a source of considerable annoyance, the barring of scrap from rails from beneath or the adjusting of sprays should be done between casts, when iron is not being poured.

The type of infection represented in accident 24 is not uncommon among men making frequent use of bars. It usually develops from the formation and breaking of a common water blister. Special mention should be made to the men of the liability of lost time if ordinary water blisters are neglected or unskillfully opened.

LADLE HOUSE.

1. Ladle man attached chain hook to rim skull. As chain tightened, finger was caught between hook and scrap. D.

2. Man hooked chain to scrap and, not noting that chain was about leg, gave hoist signal. Leg was caught and broken. C.

3. Ladle skuller, in dumping rubbish bucket, had finger caught and crushed. C.

4. Ladle cleaner signaled to raise bucket of scrap. As bucket was hoisted, it swung around and knocked man against ladle truck. B.

5. Laborer in car unloading scrap from bucket was struck by bucket swinging; broken arm. B.

6. Ladle man was signaling crane-man to lower thimble into thimble frame of cinder truck. Kept fingers on frame, and they were crushed as thimble slid into frame. B.

7. Man standing under crane was struck when chain was pulled from beneath ladle skull and swung; fractured jaw. B.

8. Ladle skuller was cleaning ladle when rim skull fell out onto foot. B.

9. Casting-machine man was cleaning lining from ladle. Ladle was turned up and the lining collapsed and fell on him. D.

10. As laborer was raking cleanings from ladle, skull on ladle nose fell on his foot. D.

11. Laborer was sledging scrap when piece bounded and struck ankle. D.

12. Iron pourer was loading scrap and tore nail from index finger by catching it on scrap. D.

13. Ladle man was prying scrap off ladle spout when bar came up suddenly, bruising groin. C.

14. Ladle liner had bar under piece of scrap which was being hoisted. Hitch slipped and scrap fell back on bar, throwing it against man's jaw, fracturing it. B.

15. Bricklayer, working about ladle pit, slipped and fell into it. D.

16. Conductor was making cut on ladles while men were cleaning ladle into quenching pit. Explosion occurred in pit, throwing hot water and metal over conductor. D.

17. Ladle-house man was pulling cleanings into chills, when iron drainings exploded. Eyelids and face burned. D.

18. Pig-machine helper was chilling ladle cleanings with hose and was cut and burned by explosion of hot metal. D.

19. Ladle liner turned furnace gas into ladle to dry lining. He then dropped burning waste into ladle and gas exploded, burning him. D.
20. Torch exploded, burning men working on ladle lining. B.
21. Laborer had piece of hot ladle scrap fall into glove. D.
22. As extra man was cleaning ladle, spark flew into eye. B.
23. As laborer was cleaning ladle, molten iron dammed in ladle by kish ran into shoe. B.
24. Ladle cleaner stepped into quenching pit filled with scalding water. B.
25. Crane operator was burned by fuse blowing out on switch. C.

The two "hooking-on" accidents are typical. Personal carefulness will prevent such injuries, as there is little or no hazard in such operations if the workman knows of the possibilities of danger and exercises reasonable precaution.

Short bars should not be used in skulling ladles, until rim skulls and loose brick have been knocked or pulled out, as they are apt to fall, when men are close up working on bottom skulls or daubing the nose of the ladle.

A safeguard to prevent accidents like No. 15 might be provided. The ladle pit is, as a rule, set with its top flush with the ladle-house floor. By extending the wall of the pit and providing a coping 18 or 13½ inches thick by perhaps 2 feet high, some protection would be afforded and no handicap to relining would be interposed. Ample light over these lining pits should be provided.

Accidents such as 1 to 15 can not be much reduced, even by the making and enforcing of rules, for such work is only slightly amenable to rules. Successful accident prevention must be obtained by largely arousing the interest of the men, directly or indirectly, in their work by inviting them to solve plant problems of operation and safety devices.

Quenching pits, that is, pits filled with water into which ladle scrap and kish are dumped are largely coming into use. Several explosions have occurred, not necessarily explosions resulting in injury, though of some violence. The reason for the use of the pits is that contact with water causes the cleanings to granulate, so that they can easily be forced through an ordinary ore-bin door, whereas the solidified skull of iron and graphite obtained by pouring ladle cleanings onto the ground or into chills, even after breaking, is so rough that it sticks in a bin. This advantage is so marked that these pits have undoubtedly come to stay. Consequently, they should be made as safe as possible; a large volume of water is necessary. The capacity of the quenching pit should be at least equal to that of the ladle from which the cleanings are dumped. Two hundred cubic feet is a safe minimum, and some plants have pits with twice to ten times this capacity. So long as the water is in large excess, explosion can not occur when hot metal is beneath it, the chilling effect being sufficient to retard the speed of the steam or hydrogen forming reaction. However, in

spite of large water capacity, the circulation of water is usually retarded and large amounts of hot metal may be in contact with small amounts of stagnant water, because the ladle cleanings drop into a bucket, submerged in the water, which obviously retards the currents of water from starting circulation throughout the pit and may permit a rapid formation of steam in contact with the hot iron. The pit bucket should have a capacity of 150 cubic feet, and should run down into the pit on a short skip incline, be submerged in the pit, and be automatically dumped. With this arrangement, without expenditure of labor or the use of a crane, the bucket can be dumped after each ladle is cleaned, and the accumulation of scrap in the bucket can be avoided as well as the contact of a large amount of hot iron with a small amount of water. If it is necessary to grapple for the bail of the bucket and to hoist the bucket into cars by an overhead crane, there is a tendency to allow a dangerous accumulation of scrap, the bucket not only becoming full but the scrap getting near the surface of the water. The pit should be provided with a heavy steel shield behind which the men can work, because occasionally heavy cleanings may be carelessly permitted to rush into the pit and may cause minor or possibly serious explosions. Bars and rakes of the longest practical length should be used. Side railings are advisable.

Explosions of gas while it is being lighted in ladles to dry them can be avoided by dropping a bunch of oily waste or burning paper in the bottom of the ladle before turning on the gas. Explosions of torches are usually caused by setting the torch on a hot surface, scrap, slag, cleanings, or plates.

Accidents like others mentioned can be largely avoided by using long bars in breaking up hard kish and skulls in starting to drain the ladle. After rim skulls and the loose cleanings have been removed, close work with short bars and sledges does not involve any marked risk, so long as the men wear goggles or masks. The ground or chills should be perfectly dry.

Accidents like accident 1 are purely a hazard of the work.

All fuses on mill circuits should be inclosed, or put where they are not in range of men working nearby.

Cranemen and crane oilers, upon coming on turn, should make a daily inspection of their cranes for loose tools, bolts, or other material that might fall off. To locate loose nuts or bolts requires a thorough examination, and it can not be expected that such defects will always be noticed, any more than it is to be expected that men will happen to be under cranes at the precise moment when a bolt or other part drops. However, daily inspection by the crane man, with an occasional regular detailed inspection by an electrical millwright, will minimize the chance of such injuries.

Injuries caused by pinching of the hands in handling ladles, or equipment, like most injuries in which men are caught in simple machinery, are not to be foreseen. After such an accident the injured frequently can not tell why or how he got caught. If men injured in this way are rightly handled by their foremen after they return to work, they can usually be converted into careful workers and active safety men for a time. To keep them interested requires effort.

SKULL DROP.

Though a skull drop is operated at all furnaces having iron ladles, it is difficult to segregate skull-drop or skull-cracker accidents at furnaces and steel plants and to definitely apportion them to furnace operations.

1. As scrap breaker was pulling chain to release ball at skull cracker, chain pulled apart, having previously broken and been mended with wire, against orders. Cut of scalp. D.

2. Craneman, in coming down ladder from cab, slipped and fell 30 feet to ground. Fractured wrist, bruised head, knee, and shoulder, and internal injuries. B.

3. Turn man started to pull pulley block off track, but as he grasped the cable the engineer started the hoist and drew turn man's hand into block. End of little finger amputated. C.

4. Laborer had eyes blinded and burned by excessive light from arc burner. D.

5. Laborer was working at scrap drop when molten iron from a piece just broken ran into water and exploded. Burns of neck, side of face, elbow, and wrist. D.

6. Ladle-house man was helping break scrap at skull cracker. In running hoist, craneman raised and stopped bucket too suddenly and caused scrap to fall off bucket onto head and shoulders of ladle-house man. Cuts and burns of scalp and shoulder. C.

7. Laborer working in scrap yard was struck by flying piece of scrap broken off by drop ball. Loss of sight of right eye. B.

8. Laborer was loading scrap on car at skull cracker and let piece fall on foot. D.

9. Laborer helping at scrap yard hooked crane hook onto scrap bucket, and signaled cranemen to hoist. He was holding hook on clevis and when crane lifted, bucket swung, catching foot between bottom of bucket and piece of heavy scrap near pile. Cut and wrenched ankle. D.

The standardized safeguard about a skull cracker is an inclosure on all four sides, built of reinforced concrete, concrete-filled steel pipe, or timbers, with an opening for the cars. At all plants, except a few where the breaking of furnace scrap is done in a ladle or the cast house, the operator is provided with a sturdy steel-plate shield, or with a cage inclosed with screens or bars. A rope or cable is preferable to a chain for releasing the ball, and should extend outside the drop inclosure. The ball man should be protected when he trips the catch. As a further measure of protection, signs

with a white "Danger" on a red background should be placed on each side and at all approaches.

The personal element is a large factor, as the outline of accidents shows. Loading and hooking on of the crane and other hand labor offer plenty of opportunity for the cooperation of the men in reducing the number of injuries after the management has provided all reasonable safeguards around the skull drop.

GAS-MAIN SYSTEM, GAS CLEANING, AND STOVES.

GAS-MAIN SYSTEM.

1. Bricklayer's helper, working on temporary platform on gas main, fell to ground. D.
2. Boltermaker's helper fell from platform in making repairs at dust catcher. D.
3. Rigger, in bolting manhole head on gas main, slipped and fell to ground. D.
4. While handling brick on top of hot-blast valve chamber, laborer slipped and fell to pavement. D.
5. Boltermaker's helper was changing bell at dust catcher. Scaffold he had built gave way, and he jumped to ground. D.
6. Carpenter overloaded platform on which he was working, causing it to give way. Fell 8 feet. C.
7. Flue-dust man, standing on plank across top of hopper car, was cleaning downleg chute. Plank broke, throwing him to the bottom of car. C.
8. Laborer's neck was burned by hot dust while he was cleaning downleg. C.
9. Laborer was cleaning gas main. Board on which he was working sank into dust, causing burns of ankle. C.
10. As yard laborer opened bell on downleg, dust flowed over leg. B.
11. Laborer was on wall next flue dust car and, thinking dust cold and hard from watering 15 minutes previous, jumped into car, sinking foot into flue dust. B.
12. Labor foreman had men cleaning under dust catcher. Ordered men from beneath, as the bell had to be dumped. Just as the bell was lowered, foreman sprang back beneath catcher to recover shovel and was caught in shower of hot dust. B.
13. Rigger allowed wrench to fall from main; it struck top of laborer's head. D.
14. Laborer was struck on head by brick from gas-main platform. D.
15. Man was holding cutter on rivet, when head flew up and hit eye. D.
16. As man was tearing scaffold from dust catcher, he punctured his hand. Neglected to report, and infection developed. D.
17. Rigger was swinging connection into place on gas main. Shackle broke, and piece swung about, knocking man to ground. B.
18. While rigger was changing hot-blast valve, an explosion of gas in hot-blast valve chamber burned him. B.

Accidents 1 and 2 could have been avoided if wooden or rope railings had been provided on the temporary platforms. Walks and railings placed along the tops of the mains would have prevented accidents 3 and 4.

Accident 5 shows that only carpenters should build temporary scaffolds. Accident 6 should not have happened as the injured man was presumably in the position to know the strength of the scaffold.

Accident 7 is a common type in yard and track work. An inspection committee can locate the defective or inadequate planks, and it is also strictly the business of labor and turn foremen to look out for such unsafe planks.

With the exception of the two accidents last mentioned, the injuries are characteristically due to lack of care or skill on the part of the injured or their foremen. The construction of platforms with toe boards and the removal of loose material from high places whence it may fall, together with signs to warn men of work overhead, are safety measures needing emphasis.

From time to time breakage of rigging tackle has caused a serious accident. Inspection of this equipment, specification of tackle for a particular job by the boss rigger, and supervision of hitches and working positions of men in the gang by either the boss rigger or a careful and capable gang foreman are essential.

The foreman involved in accident 12 had had 23½ years' experience about blast furnaces, and must have been perfectly conversant with the work and with the chance he took. In spite of a safety campaign of five years' duration, the safety impulse had seemingly not been sufficiently awakened in him. Such accidents are baffling, for if foremen, supposedly of greater intelligence than their men, fail to comprehend the rudiments of a safety campaign but persist in taking foolish chances themselves, even on the spur of the moment, it is too much to expect men of less ability to grasp the idea, and it is manifestly wrong to discipline them for their failure to do so. For instance, this particular foreman would be in poor position to reprimand even the laborer who, in accident 11, jumped off into a car of freshly dumped flue dust.

Accident 9 illustrates a method of work that is better done by flushing water into the main with a high-pressure hose. Accident 8 could have been prevented by having the dust-bell lever provided with cables running over a pulley, enabling the men dumping the dust leg to stand away. Dust catchers should be similarly equipped. A clamp to prevent the bell from being opened by a slip or by excessive weight of dust is well warranted.

Explosions of gas in the main when hot-blast valve seats are being changed are infrequent. When this work is done the gas should be drafted back through the stove nearest the furnace between the valve under replacement and the furnace. Assuming the valve in No. 2 stove leaking and the No. 1 stove valve off for cleaning, or No. 1 stove valve leaking, the gas must then be drafted past the hot-blast valve chamber under repairs to a further stove. Occasionally when the hot-

blast valve seat is slid out the gas ignites explosively. At one plant, where a serious accident of this description occurred some years ago, the tuyères are plugged when a No. 1 stove seat must be changed.

Some plants experiencing excessive cracking and leaking of valve seats have replaced the bronze water-cooled seat with solid cast-iron or steel seats. The author does not know whether their use has proved successful. Pressed and welded water-cooled seats are successfully used.

STOVES.

1. Laborer was on top of checkers cleaning same when gas escaping from burner asphyxiated him. D.
2. Rigger was caught between bracket and stairway in construction. A.
3. As hot-blast valve was being hoisted, block broke, allowing valve to fall on man's head. B.
4. Hot-blast counterweight cable broke, allowing weight to fall on man's shoulder. B. (Accident happened in 1914.)
5. Millwright, working on scaffold at hot-blast valve, fell when scaffold broke. D.
6. Helper fell from top of hot-blast stove. No witnesses. A.
7. Laborer beneath checker arches was hit on foot by pipe falling from hands of top cleaner. D.
8. Laborer was in bottom of stove when piece of brick fell through checkers onto head. D.
9. Laborer picking up brick outside stove was hit on lips by brickbat thrown by man working inside. D.
10. Hot-blast man was struck by mixing-valve wheel falling off stem owing to loose nut. C.
11. In 1914, laborer was in combustion chamber loading brick into bucket for repairs to checkers. As bucket was hoisted, brick fell over side and fractured skull. A.
12. Hot-blast man strained side pulling on valve winch. D.
13. Stove cleaner, carrying cover plate on top of stove, had toes crushed when eyebolt broke off. D.
14. Rigger, in rolling hot-blast valve along pavement, ran valve over foot. C.
15. Stove tender slipped on pavement, straining ankle. C.
16. Stove repair man crushed hand between doorframe and thumb nut in closing stove door. C.
17. Bricklayer's helper in cleaning checkers was struck in abdomen by gas pipe he was plunging up and down to dislodge brickbat in plugged checker. Peritonitis from perforation of intestine. A.
18. Millwright burned eyes by hot dust from packing gland of hot-blast valve stem. Was packing gland with blast on furnace. C.
19. Man burned hand in screwing light into socket, socket short-circuiting. D.
20. Stove tender turned steam into pipe to blow out checkers. Hose burst and steam scalded body. B.
21. Hot-blast valve burst on account of steam pressure being too high. Steam was used to thaw out valve. Helper scalded. D.
22. Laborer was cleaning stoves, and as dust was scrapped outside stove, hot flue dust flew into his eye. D.
23. Mason's laborer, in wheeling flue dust, stepped into pile of hot dust. D.

24. Pipe fitter's helper opened door of stove to see whether gas was burning, when gas flashed back, burning his face and forearm. D.

25. Gas ignited suddenly at burner door just as stove tender stepped near to observe whether it was burning. Explosive puff blew dirt in his eye. D.

26. Clinker snapper was through for day and was standing in front of stove, when door clump broke and escaping hot blast blew him against side of adjacent stove. A.

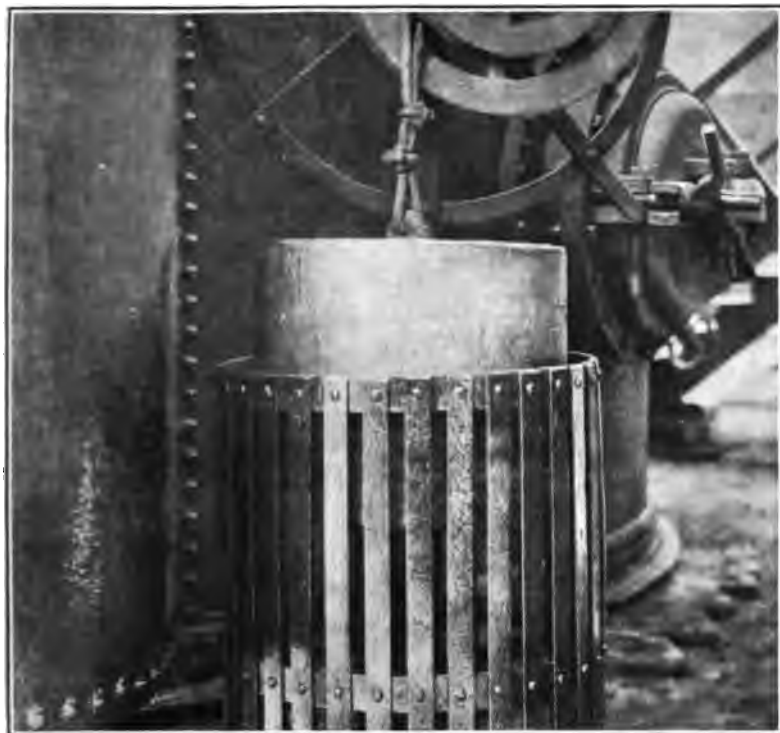
27. Stove cleaner was cleaning stove well when a slip in the furnace threw dust and flame from the bottom cleaning door. D.

28. Stove cleaner had been inside cleaning checkers, and had come out for a breath of fresh air. A slip in the furnace next to the stove caused the man to fall off the stove. A.

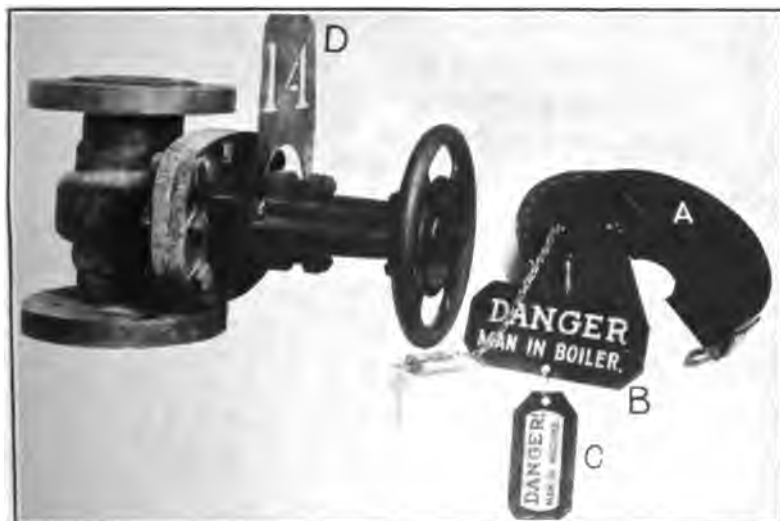
The danger of asphyxiation from gas while men are working in stoves is chiefly from leakage of gas from gas valves. Minor dangers are as follows: From gas working back into the stove when the furnace is off and the gas is being drafted back past the hot-blast valve to a stove beyond the one being cleaned; from leaky burners or mains of adjacent stoves; and from gas turned into the stove by an inexperienced stove tender. As the nose of the gas burner is within a few inches of the doorframe of the stove when the stove is off gas, and as the burner nearly always leaks slightly, it should be turned on its seat 90° away from the door, or there should be a blank put in between the burner base and the rack, or between the burner and the interior seating valve, or the stove burner door should be shut. In addition it is advisable to tag or lock the hot-blast valve, as well as the burner valve. Excessive leaks from near-by burners or downlegs should be plugged.

Stove counterweights should be inclosed. (Pl. IX, A.) Counterweights should hang close enough to the ground so that this may be done. Many counterweights are suspended extremely high, and present a difficult proposition, and auxiliary cables are practically the only means of holding these up if the main cable breaks.

Stove tops should be adequately railed, with a toe board of 12-inch plate at the foot. They are preferably made of angle iron not less than 2 by 2 inches by $\frac{1}{4}$ inch, and with at least one intermediate railing. If made 4 feet high, there should be two intermediate rails. Access to dome cleaning doors at the top and to checker cleaning doors beneath the main platform should be by railed steps if possible. When not possible, the necessity for climbing over the top of railings to gain access to ladders should be eliminated. This fault is common in many stoves. Each ladder should be arranged so that a man can step off from the top platform directly to the ladder and descend with his back to the shell of the stove. Manholes on top of domes of two-pass doors should be provided with platforms. Such equipment is necessary because of the great height of the stoves, and because of the possibility of men being partly gassed and dizzy.

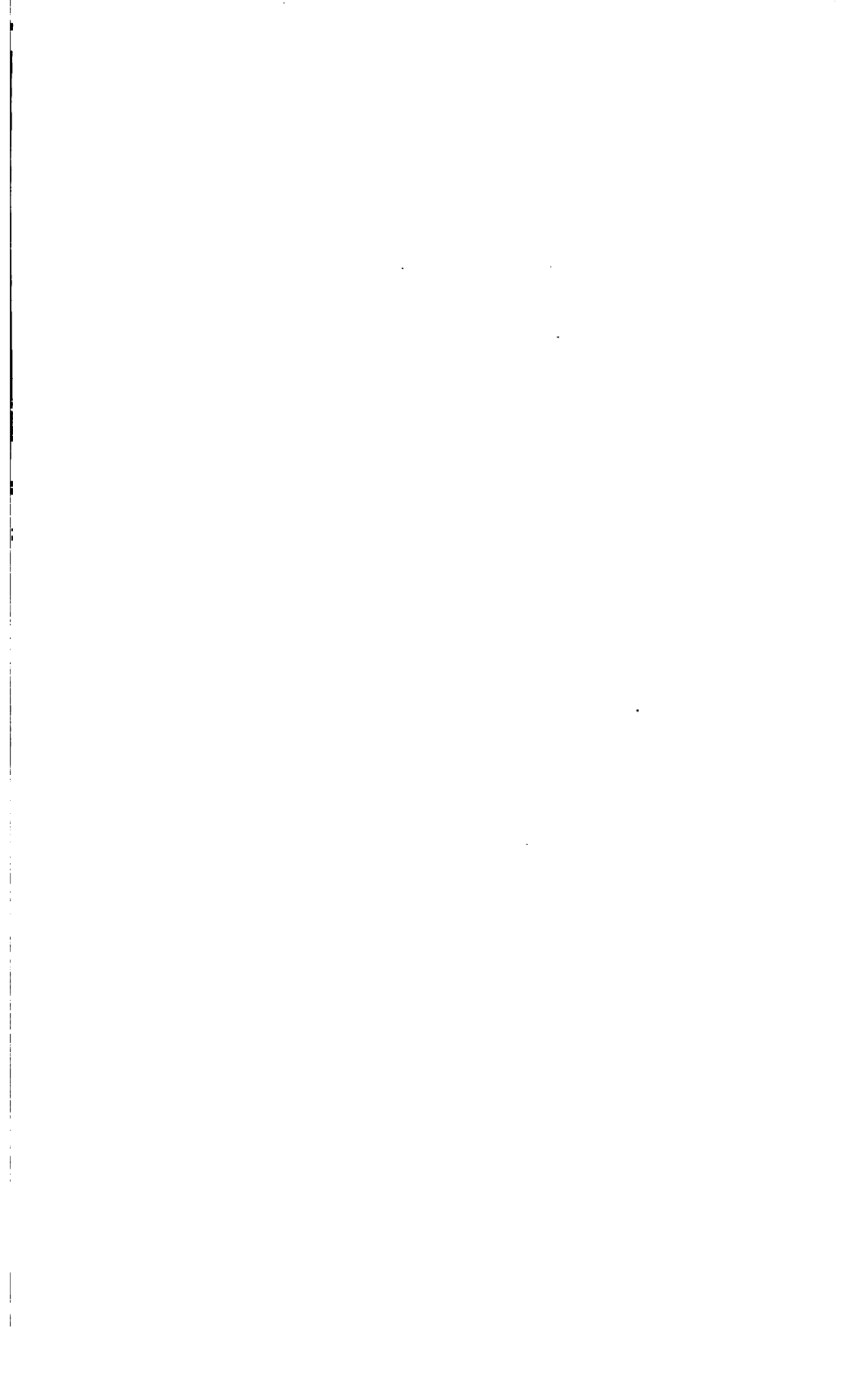


A. GUARD FOR STONE COUNTERWEIGHT.



B. VALVE LOCK AND DANGER TAGS FOR BOILER.

A, Hinged metal case, painted red, for locking valve stem; B and C, danger tags; D, boiler number.



Railed platforms at the hot-blast valve seat and at the top of the hot-blast valve chamber are standard construction on newly erected plants. If these have not been installed, a collapsible scaffold support of light angles and flats, with plank to furnish the flooring, is a time-saving device, and safer than a carelessly erected platform.

The danger of injury from brick falling from the top of the stove through the checker openings is practically eliminated if the stove cleaners enter the top first, open and brush all checkers, dislodge disintegrated and loose checker brick, and come out before men go under the checker arches to remove flue dust and brick debris. In tearing out skin walls and brickwork, liability of injury from tumbling or flying brick from the doors is minimized by hanging a steel plate a few feet in front of the door.

The mention of fatal injury is made because this kind of accident was formerly not uncommon during the relining of furnaces and stoves. It is almost universal practice now to build a substantially covered scaffold about 7 feet high in the bottom of the combustion chamber when a skin wall is being built. There is an opening provided for the brick bucket. As a rule, brick for repairs to the top of the checker are hoisted from the outside of the stove and offer little possibility of injury aside from handling.

Gloves should be worn when one is screwing electric-light bulbs into sockets.

Only approved wire-wrapped three-ply steam hose, purchased as such, should be used for blowing out stove checkers or boiler tubes, and it should be tightly clamped onto a steam union with corrugated pipe extension inside the hose.

As a rule, hot-blast valves and seats, as well as other bronze furnace-cooling equipment, are designed for use only with low pressures. High-pressure water or steam is, therefore, to be used with caution.

Flue-dust injuries about stoves are, of course, eliminated if washed gas is used. In wheeling dust, care should be taken not to fill the barrow too full. Stepping in dust is well known to be dangerous, and injuries from this cause are due to carelessness. All dust should be promptly cleaned up. Wetting it is even somewhat hazardous, for if a heavy stream of water is turned on it, it will fly surprisingly.

Stove cleaners should always wear goggles.

Every man should be instructed not to open stove doors, nor to go near them, when a stove is on gas. Only stove tenders and cleaners should be allowed to open the doors, for they are more apt to know whether the gas is liable to flash on account of excessive gas, dirty checkers, or poor drafts.

It should not be necessary for an experienced stove tender to look into a cleaning or air door to see whether the gas lights, as it will

either light promptly at the nose of the burner, back in the well, or if it lights only at the top of the well there will be a puff. If the gas does not light promptly, it should be turned off at once and a hotter fire of waste should be provided. When the stove is full of gas a very explosive ignition may take place.

Other characteristic injuries from slips about stoves are: Broken leg from jumping off hot-blast valve to escape slips, sprained wrist from fall while running to escape a slip, and bruised head and back from material from slip.

If furnaces do not have closed tops, a signal whistle at least 2½ or 3 inches in size should be placed at the top of the trestle. It should have operating wires for use by the stock-house men, who usually know when a furnace is hanging, and for use by the furnace blower, who can usually determine closely when the furnace is about to slip if the blast is not checked, or, if he has to check the furnace, can give ample warning.

If the furnace is slipping badly and more than one check is necessary, time should be given stove cleaners to come out of the stove because gas may draft back into the stove. A stove cleaner should come out from a cleaning door away from the furnace, and to avoid uncertainty the door facing the furnace should be marked on the inside with red chalk, as confusion is easy inside the stove. Stove-well cleaners should be warned away from cleaning doors also. It is safer to clean the wells with the burner shut off, as otherwise there is likelihood of a burst of gas when the furnace slips. The warning whistle is the only whistle necessary about the furnace stack, and even where the blowing-room whistle is audible all over the plant, a special warning is essential.

One hazard about stoves is the breaking of bolts or door yokes when the stove is on blast. Such breaks occur without warning; consequently, men must not be allowed to stand in front of stove doors. Neither must stove tenders be allowed to tighten nuts on air inlets, or on burner doors to stop air leaks while the stove is on blast. It is equally dangerous to loosen or partly open blow-off valves before closing the cold-blast and the hot-blast valves. The practice of excessively tightening thumb nuts on various doorframes before putting a stove on blast is almost as bad as tightening them after the blast is on, for as the doorframe and the door heat up when the blast is on, an increased strain is put on the bolts which, if the elastic limit is repeatedly exceeded, will ultimately fail. Cast-steel door yokes are in some plants replaced with two small structural-beam sections, which are considered much more homogeneous and less liable to conceal hidden defects such as blowholes or segregations.

A locking device for valves about a boiler is shown in Plate IX, B. This device is used as a protection to a man entering a boiler. At

one plant when a man enters a boiler two steam valves, two feed valves, and the blow-off valves between the boiler and the main headers are each closed and securely locked by the device (A) shown, which is a hinged metal case, painted red, that is slipped over the valve handle and padlocked. A metal tag, B, reading "Danger, man in boiler," is attached to each case. A small tag, C, reading "Danger, man on machine," is attached as an additional precaution. All boilers are distinctly numbered both in front and in rear, the crown valve on top of each being numbered correspondingly with a metal plate, D. This number can not be effaced and prevents possible confusion leading to the operation or locking of the wrong valve.

GAS-FIRED BOILER HOUSE.

1. Boiler tender was mudding up clean-out doors on dust leg and stayed in gas too long. Asphyxia. D.
2. Water tender, in changing gas burner, was overcome by gas. D.
3. Helper was working on top of boiler setting and was overcome with gas. D.
4. Water tender, in trying to stop leak in gas burner, was overcome with gas. C.
5. Fireman was in combustion chamber cleaning it. Became gassed and fell into hot clinker. B.
6. Fireman was found asphyxiated on top of boiler setting. A.
7. Boiler-house laborer was inside setting cleaning dust out and was asphyxiated from gas from leaky burner. A.
8. Carpenter working on scaffold in boiler house was overcome with gas and fell to floor. A.
9. As water tender was throwing coal through fire door, gas flashed out of furnace. D.
10. Boiler cleaner was blowing tubes, when furnace slipped, causing flame to blow out. C.
11. As scaler was cleaning setting beneath tubes, dust fell onto hand and arm, burning them. D.
12. Foreman put high-pressure hose on flue dust, causing it to fly and burn arms. D.
13. Boiler cleaner stepped into flue dust inside boiler. B.
14. Foreman stepped on plate over ditch. Plate tipped and leg went into hot water. D.
15. Scaler was cleaning tubes when the blow-off valve was opened, allowing steam to burn scaler. D.
16. Fireman had cleaned fire and pulled ashes from ash box. In spraying ashes nozzle came off hose and the steam suddenly generated burned him. D.
17. Fireman was cut by bursting gage glass. D.
18. Water tender was scalded and cut by bursting water glass as he turned water in after replacing defective glass. B.
19. Pipe blew off steam hose, scalding man standing beneath. C.
20. Water tender was scalded by bursting tube while blowing off dust on tubes. B.
21. Water tender fell into ditch containing scalding water beneath blow-off line. C.

22. Water tender, not being able to see on account of escaping steam, fell into uncovered blow-off ditch, lacerating leg. B.
23. Laborer was on steam line between boiler house and engine house, painting, and fell, breaking hip. B.
24. Ladder slipped, throwing boiler cleaner to the floor. C.
25. Laborer, working on top of boiler, made a misstep and fell to pavement. B.
26. Millwright fell down steps from top of boiler setting. C.
27. Fireman was firing boiler while men outside were throwing coal onto pile inside house. Lump struck leg. C.
28. Man was struck by board falling from roof truss. D.
29. Man was moving portable scaffold when plank fell off trestle onto foot. D.
30. Eighteen hand-labor accidents of the same characteristic type repeatedly presented occurred, causing disabilities only four of which were in excess of 14 days. Lack of skill or caution was a contributing factor in each.

The less serious accidents from gas were, with one exception, to men outside the boiler setting and were due to gas leaks about the clean-out doors or burners. However, the chief cause was leaky burners, these leaks as a rule being developed by the abrasion of the iron by the sharp coke or ore dust carried in the furnace gas. As soon as a hole is worn through a burner, especially if the seat becomes eroded and gas starts to escape, the burner should be changed at once, for claying up or cementing the hole is only a temporary expedient and the gas will soon be escaping again. Thus, not only is the boiler-house force exposed to gaseous atmosphere continually, and is from time to time working in gaseous air while stopping the leaks, but the gas drifts up over the top of the boiler setting, and about steam valves where work is occasionally undertaken by other members of the furnace force. Previous records disclose numerous falls from pipes, ladders, and boiler tops as the result of partial asphyxia. Keeping boiler burners tight is a requisite for the safety of the boiler-house force.

Asphyxiation of a man inside a boiler setting is always due to a leaky burner, usually of the boiler in which the man is working but sometimes of adjacent boilers. When the slide burner is used, a plate should be placed between the nose of the burner and the burner opening in the setting. Single-valve burners are dangerous because the bell or the seat may be cut by flue dust sufficiently to allow dangerous volumes of gas to drift into the boiler setting from the outside. When a single interior sealing valve is installed, some provision is almost imperative for blanking off the burner between the valve and nose of the burner. If the burners can be withdrawn from the setting, a plate may be inserted.

The best interior-valve burner is one having two valves, between which a handhole or other door may be opened to the air. Thus any gas escaping past the first valve may be diverted into the house instead of going to the interior of the boiler setting. A company

operating several furnaces removes the burner bodily and clamps on a blank whenever men are required to go inside either boilers or stoves. Many companies require the burner valve to be locked shut. This practice is commendable, but does not eliminate minor leaks, which are the major cause of asphyxia inside settings. Rigid inspection and elimination of small leaks is essential. For changing leaky burners, inserting blanks into the gas-main side of the valves, or other occasional necessary work in gaseous atmosphere, a half-hour breathing apparatus is useful and is being widely adopted.

Special attention should be given to good illumination in the boiler houses. High-voltage electric lamps are standard, and in the absence of overhead mains that interfere with lighting the floor, they are preferably hung so as to cast some light on the tops of the boiler settings. When they are of necessity placed lower, it is advisable to have a light at each of the stairways leading to overhead walks, and in large plants two or more intermediate lights should be placed at the top.

Sufficient men should be assigned to boiler cleaning so that one can remain outside and give assistance if the man inside is gassed. The draft regulator or valve should be left open wide enough to induce a perceptible current of air, and doors in the front of the setting should be closed when adjacent burners are leaking, the work being done on the side cleaning doors. Men should be repeatedly warned about the danger of gas about burners, in side settings, and on top of boilers, as experience frequently dulls rather than sharpens any disposition toward apprehension of serious effect from gas. Most cases of "gassing" do not result in anything more serious than a severe headache for a few hours, involving no loss of time, and as a rule serious gas accidents at any one plant occur only at long intervals. Thus, the danger from gas is not always apparent to men even of long experience. The length of service of the men concerned in the accidents outlined above was 1 week, 4 and 5 months, and 2, 3, 4, 5, 6, 10, and "several" years.

A final cause of asphyxia is leaking explosion doors in gas mains inside boiler houses. The boiler-house gas main should be outside the house whenever possible. In 1914 one man was fatally overcome in attempting to close a door blown open by a small explosion in a gas main inside the boiler house, another man attempting rescue was also overcome. When the main is inside the house and equipped with easily lifted explosion doors, there is an almost continued leakage of gas, which accumulates about the top of the boilers, where the steam and other valves are usually situated. Such doors can safely be bolted down, and reliance placed in doors outside the house.

It is manifestly not feasible to place railings about the top of each boiler setting. One of the falls from the top of a boiler re-

sulted from deliberate foolhardiness. The work resulting in another fall would in all probability have necessitated the removal of a railing, if present. Walks from boiler to boiler should of course be railed, and it is important to have platforms at individual valves at the main steam header, or on overhead by-passes to which access is difficult. The handwheels on these valves, if of the inside screw type, should be secured with lock nuts. Some few plants have provided rails at the outside edge of the "dog house" or fore oven.

Whenever a ladder is set on an iron plate, someone must hold the base of the ladder. There is no known ladder base that will absolutely prevent slipping on smooth iron plates. It is, of course, desirable that floor plates, ditch plates, and manhole covers be cast with a checkered surface. Several plants have done away with the use of ladders in blowing boiler tubes by providing permanent platforms at the tube cleaning doors.

Falls down steps are largely due to carelessness. Plant managements can only minimize the probability of these accidents by careful construction of the stairs. (See Pl. X, A.) The following features are suggested: Nonslip treads, rails on both sides, treads not less than 9 inches wide, risers of such height that the sum of riser and the tread is not more than $17\frac{1}{2}$ inches.

Ditches for blow-off lines should be covered. If changes in house equipment are necessary to permit covering the ditches they should be made.

Another hazard, not disclosed in accidents in 1915, lies in lighting the furnace gas. There is no danger in this work provided a hot fire is built on the grates before the gas is turned in. When, however, gas from a newly blown-in furnace is to be lighted at the furthest boiler, it is also essential to place a very large amount of burning oily waste about the nose of the burner. Occasionally this gas is tardy in lighting and fills the setting before it ignites, when it catches fire with explosive violence. Men should be kept away when new gas is being brought down; the gas should be lighted only under the supervision of the furnace superintendent or foreman, and if ignition does not take place promptly the burner should be shut off at once and additional time given for the main to fill and push out air. Filling the boiler with gas is to be avoided—the gas should light easily and gently at the nose of the burner.

An explosive burst of flames from a boiler in which the gas is ignited and burning satisfactorily is exceptional. It occasionally happens when the gas is of the "calico" variety or "too hot to burn." It also occurs when the furnace slips. A whistle or light is sometimes installed to give warning of impending slips, but, as a rule, the blowing-room whistle is clearly audible in the boiler room and is taken as indicative of a slip. Unfortunately on single furnaces this whistle



A. CONCRETE STAIR TREAD. INVERTED CHANNEL IRON FILLED WITH CONCRETE WHICH HAS A ROUGH SURFACE OF HARD CINDER.



B. PLATE GUARD TO PREVENT MEN STANDING BETWEEN BUILDING AND HOT-METAL CAR TRACK; ALSO GUARD RAILING AT CORNER TO PREVENT MEN STEPPING OUT ONTO TRACK.

is also a signal to fire coal to tide the pumps and power equipment over a possible 10-minute shutdown. At other times when a furnace is slipping, it may be so "tight" that little gas comes over to the boilers, again requiring another firing of coal. Curiously, if a slip is of sufficient violence to introduce a hazard of burning flames shooting out over a fireman there is, just prior to the slip, a momentary slacking of gas at the burners. To experienced men this sudden cessation is significant and gives them time to dodge back before a rush of gas comes. Men engaged in blowing tubes, however, are usually on ladders and can not as easily escape and should, therefore, get away from the doors at once upon hearing a check blown.

To prevent accidents from hot flue dust in boilers, it is necessary to dislodge all loose dust with a steam hose and to wet it thoroughly with a pressure hose before entering the setting. Men should not be allowed to go inside with hose to wet the clinker but should do the wetting from the outside.

Loose boards, pipes, or defective and loose roofing should be taken down from overhead. Unless a rule requiring removal of such material is rigorously enforced a surprising amount will accumulate, being left after occasional repair roofing jobs. Every plank on portable scaffolds should have a bolt dropped through a hole at the end to prevent the plank from sliding off the supports.

Many serious and fatal accidents like No. 15 have occurred. The only assurance of safety is to lock the blow-off valve when the boiler is taken off for cleaning. In addition to the regular boiler blow-off valve, each blow-off pipe should have a straightway or stopcock next to the boiler to prevent back pressure on the header from lifting the seat of the blow-off valve. If this is not installed, a blank flange should be placed between the blow-off valve and pipe flanges. To relieve pressure in the header, there should be a T in the blow-off line for each battery of eight boilers, permitting a pipe to be run up through the roof; or a similar pipe should be installed at the hot well.

Permanent gage-glass guards are not much in favor because they obstruct or interrupt a view from the boiler-house floor. When there is a special platform or walk for the water tender, this objection has no weight, and guards of wire glass should be provided. If no fixed guards are provided, gage glasses away from the usual or necessary travel should have a swivel guard that can be rotated around the glass while work is being done on the water columns, such as inserting new glasses.

A new gage glass must be inserted correctly, as the bursting of a gage glass may be due more to improper packing and the glands not being in line than to defects in the glass. Care must be taken that water-column valves are in alignment so that the glass can expand

freely. Glands should be screwed down tight enough to set the packing and should then be loosened slightly so that the glass can be turned with the fingers. There will be a slight leakage of steam which soon stops as the glass becomes heated. The glass should not bind the gland. In turning on the new glass, the water valve should be opened, first slowly, until the glass is filled, when the steam valve may be opened. For closing valves because of breakage, a desirable safeguard is the use of pendent chains or rods on valves operated by levers.

The number of accidents such as 19 would be lessened by the use of three-ply wire-wrapped steam hose and of a strong, tight clamp for clamping the hose to the steam pipe, and by weekly inspection of the equipment by the boiler-house foreman.

BLOWING ROOM.

1. Assistant engineer was in basement filling water-seal valve and was overcome with leaking gas from engine intake. Foreman in rescuing him was also overcome. D.

2. Oiler in basement getting supplies of oil was overcome with gas. D.

3. Oiler working about inlet valve of gas engine, which was shut down, became gassed and fell. C.

4. Repair man was passing under power-house crane when craneman ran blocks all the way up, breaking cables and allowing blocks to fall on man's back. D.

5. As millwright was putting casting on "spares" pile, craneman let casting down on his foot. D.

6. Electrician on crane repairing it neglected to pull safety switch. Crane-man moved crane or trolley and amputated end of finger. D.

7. Laborer was dumping bucket and caught fingers between bail and bucket. C.

8. Painter glazing windows from swing scaffold was told to raise scaffold to allow crane to pass. Through misunderstanding painter lowered it again; crane upon return struck scaffold and threw him to floor. B.

9. Lineman was painting conduits when rung of ladder broke, throwing him against switchboard, where he received burns from contact with live part. D.

10. Oiler in cleaning incandescent globe was burned by globe breaking. D.

11. Electrician was cutting socket off end of electric-light wire when current short-circuited, burning palm. Had not turned off current, though switch was within 3 feet of work. D.

12. Oiler was wiping inlet-valve mechanism and valve hook crushed finger. C.

13. Wiper was caught by rocker arm of air valve on top of air tub. D.

14. Oiler was struck on head by ball governor. C.

15. Oiler was caught between reach rod and wrist plate of valve gear on blowing engine. C.

16. Oiler was screwing down grease cup and was struck by web of crank shaft. C.

17. Pump engineer was wiping service pump and scratched hand. Did not report, and infection developed. C.

18. Machinist, in carrying sheet iron past flywheel in blowing room, in some way caught sheet in wheel, twisting it against wrist. D.

19. Oiler was using bar to move flywheel to get crosshead in position. Engineer turned on steam to assist, causing quick throw of wheel to fling bar against oiler. D.

20. Pump engineer caught finger in link of valve gear of pressure pump. D.

21. Engineer caught sleeve on cam of gas engine while adjusting igniter, amputating thumb. B.

22. Second engineer was wiping eccentric rod and caught hand between eccentric strap and main-bearing cap. B.

23. Blowing engineer put hand on crosshead to feel whether there was any looseness and caught hand between crosshead and bottom of cylinder stuffing box. D.

24. Engineer was on bed frame of condenser engine to fill grease cups on eccentric, when foot slipped and he was struck by crank. D.

25. Laborer was washing hands under the drip of exhaust pipe on auxiliary boiler wash-out pump, when pump was started. Scalded body, thigh, and arm. C.

26. Pump tender, while in basement with torch, fell and spilled burning oil over hand. D.

27. Rigger, in repairing engine, had waste stuffed in sleeve at wrist to prevent oil running up sleeve, and waste caught fire from torch. B.

28. Helper stepped into open trap door. D.

29. Pump engineer was walking on offset of wall of cellar and made misstep. Fell to cellar floor. C.

30. As engineer was wiping rods from platform, his foot slipped, and he fell onto water-valve handwheel. C.

31. Oiler, in wiping main bearing of blowing engine, fell into flywheel pit. C.

32. Craneman, in coming down ladder, missed footing and fell. B.

33. Millwright watching men drive key in gas-engine crosshead was struck when key broke and flew. D.

34. Fireman was drawing water at power-house spigot when icicle fell from roof on him. C.

35. Bricklayer on platform stepped back against brick pile, causing brick to fall on man beneath. D.

36. Foreman was blocking up cylinder head when blocking flew out, allowing casting to come down on foot. B.

37. Machinist, in putting cylinder head on pump, caught finger. B.

38. Millwright was on ladder pulling on pipe wrench. Wrench slipped and man fell to floor. B.

39. Laborer, in wheeling brick over plank runway, fell into hole when plank wobbled. D.

40. Condenser man, in walking up plank to engine-room door, stepped on nail in board. D.

Gas about gas-engine basements comes largely from imperfectly closing gate valves employed to shut off the gas when the engine is shut down. The seats of the valves become filled with a deposit of fine fume which prevents tight closure. In addition to such a valve there should be a water seal valve, between the gate valve and the inlet valve, which can be filled immediately upon shutting down. In addition, the air for the engine mixing valve should be taken from the outside through a main up to the mixing and inlet valve, exactly as the gas is led in. With this provision any gas leaking past the gas valve is led back to the atmosphere.

Waste-water pipes from the engine jacket and piston rod should be trapped to prevent gas backing into the basement from sewers carrying water from gas scrubbers.

Vibration of pipe or depreciation of gasketed flanges may also allow the escape of dangerous volumes of gas, and it is generally considered most advisable to provide forced ventilation in the basement. The best practice is to force air from outside the building through the basement and along the base of the engines, thence up about the bed plates into the engine room, and out through the roof ventilators. One engine room housing four gas engines uses for this purpose a 42,000 cubic foot displacement fan, which is calculated to renew the air in the basement completely every two minutes.

Every gas-engine-house crew should have instruction in the Schaefer method of resuscitation from gas and electric shock. Not less than two men should go into the basement.

A direct current of 220 to 250 volts is commonly used in mills for lighting and driving auxiliary motors. However, at steel-mill furnaces, an alternating current of 6,600 volts is common. This is transformed at one or more substations to low-voltage direct current. The only furnacemen exposed to the high current are switchboard attendants and repair men in the engine and power room. Linemen are sometimes under the necessity of working near such a high-voltage line when it is carried on poles that also carry low-voltage or telephone lines. Men should never work on lines carrying more than 440 volts unless the circuit is open. They should be sure that the circuit is open at each point from which the line is supplied with power and all three wires of a transmission line should be grounded and short-circuited on each side of a point where work is to be done. The handling of high-voltage equipment is a specialized field and none but experienced men under the direction and approval of the line foreman, or chief electrician or assistant, should do any work about such equipment. High-voltage wires should preferably be placed in underground tunnels or conduits, where there is no possibility of their becoming crossed with low-voltage wires.

Work on low-voltage systems is not dangerous if precautions are taken. When possible the current should be turned off. It is important to realize the difficulty of insulating the body so that a circuit can not be completed through it. There is no perfect insulator, though for 250-volt current a number of materials may be so considered. Floors at switchboards are usually provided with rubber mats, but for other places a dry board, free from nails, may be used. Rubber or leather gloves without metallic fastenings give protection from shock. Thin or cracked gloves, or leather gloves damp with water or perspiration, give no protection. The use of rubber tape on pliers, screw drivers, or wrenches is helpful. The

tape should cover all the metal except that part essential for work, because there is always a chance of the tool slipping or of the fingers touching an unwound part. When a workman has merely to make an adjustment he should, if possible, use only one hand.

None but experienced men should work on lamp circuits, and it is to be noted that standing on insulating material will not prevent shock or burns from short-circuited lamp sockets, exploding bulbs, or arcs between charged wires when the insulation has worn off or the cord is cut. A workman changing or cleaning incandescent globes should wear gloves, and should turn his face away from the globes. Drop lights should not be used for extension lights, but a standard weather-proof socket, attached to packing-house cord, should be used.

Safeguards would have prevented most of the falls. For instance, a trapdoor or a manhole cover can often be constructed so that three sides of the hole will be guarded when the door or the cover is up. Otherwise a portable guardrail may be placed about the hole. Railed platforms can be provided along places where it is necessary for men to walk to attend to equipment. Falls from platforms, cylinder heads or bearings can be prevented by railings, toe boards, or steps with standard safety treads. Ladders to cranes may be provided with cages.

An occasional cause of injury is falls on oily platforms or steps. It is, of course, most difficult to keep some places clear of oil, and the most that can be done is to provide checkered plates and safety treads on the steps and to prevent excessive oiliness.

Four injuries were caused by the presence of men working overhead. In construction men must often be working above other men, but in regular operating this hazard can usually be avoided. If men must work above other men, the scaffolds, platforms, or places of work should be adequately provided with high toe boards. If these are impracticable, effort should be made to shift to some other place the men above or the men beneath for the time being.

Men should be warned of the danger of flying fragments from rivets, spalls, or breaking keys when other men are sledging or cutting on steel or iron equipment.

Some one man should be given the duty of knocking down icicles. Buildings like offices, engine rooms, and shops can be equipped with steam exhaust pipes under the eaves which will prevent the formation of dangerously long icicles.

The waste of time in accidents is large. If to the time lost by each injured man were added the time of men diverted from their work to assist injured men to the hospital and the time consumed in their return and in their talk about the accident, the total loss of time from useful work would be almost unbelievable. The experience of large

companies indicates that injuries from hand labor and hand tools are decreased with the greatest difficulty, although generally the real reason for the persistence of such accidents seems to be that little systematic training is given in the moving of materials, the handling of heavy weights, in piling and loading materials, or in using simple hand tools, as the newest men are generally put to work at this rough or so-called unskilled labor. To the man in the labor gang there is presented nearly every week a job to which he is new. Perhaps no two men in the gang do the work alike. Often before he has learned the best way to do one kind of work he may be shifted to other work or may be hurt. To assign a man a job of wheeling bricks with a caution to be careful but without instruction in loading and piling is somewhat like giving him a fuse, detonator, and stick of dynamite to set off, and cautioning him to be careful but giving him no definite instructions. That skilled laborers performing work rated as unskilled sometimes get hurt only substantiates the significance of the usual desultory manner of doing the work.

Many of accidents Nos. 11 to 22 could have been prevented by suitable guards about the governor, air valve, or cam, such guards being standard equipment. All the men injured were English-speaking, 11 being Americans with experience of 2 days, "few" days, 1 month, 2 years (2), 3 years, 5 years (2), $7\frac{1}{2}$ years, and more than 10 years (3).

Rules have been made prohibiting the wiping of engine parts while the engines are in motion, but such rules do not apply to engines in a blowing room because the blowing engines, pumps, and auxiliaries must be kept in operation. Although such engines are operated at slow speed, rarely making over 40 revolutions per minute, this relative slowness is often deceptive. Too much stress can not be laid on the importance of carefully instructing new men as to inherent dangers, and of bringing typical injuries to the attention of the blowing-room force so that continual work about this equipment will not cause them to become careless.

The provision of ample light is practicable in blowing rooms, if anywhere. Extension lights for use when additional light is required should be provided. The use of the common hand torch should be cut down as much as possible.

SHOPS.

1. Laborer placed chain about bell rod and signaled crane to hoist. Kept hand inside sling and got thumb crushed. B.
2. Machine man was screwing incandescent globe into socket when it broke, cutting and burning hand. D.
3. Machinist's apprentice burned fingers in pulling plug from socket. D.
4. Repair man put up temporary scaffold to repair belt. Scaffold collapsed and man fractured skull. B.

5. Millwright was pouring babbitt when explosion occurred. Face burned. B.
6. Crane runner was pouring babbitt into cable socket when babbitt exploded. Burned face and neck. D.
7. Machinist was hit by wheel from overhead crane. D.
8. Machinist's helper's toes were crushed by valve yoke, which was knocked off bench by fellow workman. C.
9. As machinist's apprentice was holding iron bar for blacksmith to cut it, a piece fell on his foot. D.
10. As blacksmith was bending steel under steam hammer, a piece flew into his eye. C.
11. Molder, breaking up tuyères, was burned by piece flying inside of buttoned shoe that did not fit the ankle. D.
12. As machinist was chipping casting, a chip broke his goggles, cutting eyelid. D.
13. Carpenter foreman was turning wood on lathe when wood flew out and struck face. D.
14. Machinist was turning casting in lathe; sleeve of jacket caught in lathe dog and bruised arm. C.
15. Machinist's helper was shifting belt by hand and slipped, striking face against belt. D.
16. Machinist's helper caught arm between pulley and belt in shifting belt. C.
17. Machinist was putting dressing on belt when hand was caught and pulled between belt and motor pulley. C.
18. Machinist was grinding tool, which was jerked, causing finger to catch between tool rest and wheel. C.
19. As blacksmith's helper was grinding burrs from bar, his thumb was jerked against emery wheel. C.
20. Molder had splinter in finger and had fellow workman remove it. Finger became infected, and he had his own doctor treat it. Finally, after four months, he reported to works hospital, when finger had to be amputated.
21. Helper was moving casting. It struck and crushed lighted torch, and flames burned arm and face. D.
22. Accidents from hand labor and hand tools occur in the shop as elsewhere, typical causes being falling ladders, foot caught, falling or dropped material, and misuse of tools.

When men are handling bulbs or working on electrical equipment about furnaces, the wearing of gloves will prevent many injuries from electricity.

Common sense in the use of switches would have prevented accident 3.

Forms into which Babbitt metal or lead is to be poured should be entirely free from moisture. They should, in addition, be heated so that they are perceptibly warm to the touch. Cold iron or steel, though seemingly dry, has sometimes caused explosion. Warming can be done with a slight fire of kindling, with glowing coals or coke, or with burning gas. If a form has been prepared and then left standing before the metal is poured careful examination should be made to see that the form is dry. A mask of fine-mesh wire is an effective protection for a man pouring babbitt. This is similar to a welder's mask and covers the entire face. Goggles cover the eyes but do not protect the face.

Overhanging gears, or split gears, with the exception of armature pinions, should not be used on cranes. If the use of such a gear can not be avoided, the shaft should extend beyond the hub of the gear for a distance equal approximately to the diameter of the shaft, and the key should be locked in place to prevent the gear from working off the end of the shaft.

Goggles used in chipping or in work about steam hammers, etc., should be selected with especial care. The lenses should be the strongest obtainable, and should be set in deep-grooved rims so that they can not be driven through. One accident of the type of accident 12 will bring goggles into disrepute, when the many cases in which injury is prevented will get little if any publicity.

Many injuries to the toes can be avoided by furnishing shoes at cost through the storeroom. Certain manufacturers are specializing in especially stout shoes with rigid box toes. These protect the wearer's toes from injury from falling planks, bars, moderately heavy plates, or small castings which ordinarily cause painful injuries.

In regard to emery wheels, clearly it is not the much-discussed bursting wheel that causes accidents, but wheels that have been unevenly ground down, are out of true, or have poorly adjusted tool rests. These wheels should, of course, have tapered sides and large heavy safety collars, and should be provided with hoods. Such wheels offer little danger from bursting; the hazard comes from unskillful use.

Specialists have suggested many safeguards for machine shops, such as the use of safety set screws, safety collars, belt guards, special devices for wood machine tools, emergency stops, and provision for oiling line shafting, with which furnace managers, foremen, and machinists are now largely familiar. Bringing of this equipment up to present accepted safety standards is an easy way of reducing accident-insurance costs. Automatic belt shifters, safety dogs, and guarded gears and transmissions would have prevented many of the accidents mentioned, though it is to be noted that even with machine tools, which can be most fully guarded of any equipment about furnace plants, accidents due to inattention persist.

YARDS.

1. As rigger was pulling board up on scaffold, hitch slipped and rigger lost balance and fell. B.

2. Rigger, in attempting to fasten snatch block on jack pole while it was being hoisted, caught finger. C.

3. Man was guiding rope on niggerhead of hoist. Rope slipped, and as man was getting turn on safety cleat, line got away and became tangled, and man was drawn into niggerhead. B.

4. Laborer held rope too long and had hand cut on niggerhead. D.

5. Laborer had hand on casting being hoisted, when sling slipped, catching hand. D.
6. Bail of bucket fell on ashman's finger. C.
7. While machinist was making adjustments to hoist engine, engineer allowed engine to run to lower bucket. Machinist's finger and thumb cut off. B.
8. Laborer was running electric-burning machine and eyes became inflamed. D.
9. Trespasser (boy) climbed pole to 6,600-volt line and was electrocuted. A.
10. To escape a shower of coke when furnace slipped, laborer ran to shelter and cut forehead on overhead obstruction. D.
11. As second helper was walking to cast house he was hit on head by coke from furnace slip. D.
12. Laborer was hit on head by material from furnace slip. D.
13. Blower was standing outside cast house watching furnace when it suddenly slipped. Limestone lump struck him on head. B.
14. Six men were caught in a shower of material from a furnace slip, receiving bruises and burns. Four D and two O disabilities.
15. Sweeper was attempting to recover shovel from fellow workman and was knocked down in the scuffle. D.
16. Laborer slipped on piece of coal and turned ankle. D.
17. Laborer stumbled over brick and bruised shoulder. D.
18. Carpenter fell overboard, striking elbow. D.
19. As laborer was wheeling brick over depression on plank, plank turned and he fell 4 feet. Bruised shoulder. D.
20. Laborer fell off plank in walking over ditch. Bruised hip and foot. C.
21. As pipe fitter was walking down plank to yard feet slipped. Sprained foot. D.
22. Laborer fell down steps at entrance. Sprained ankle. D.
23. Electric inspector, coming down from platform, slipped and fell 12 feet to yard. Sprained ankle. C.
24. Locomotive engineer was hit by corrugated iron falling from roof. Contused back. D.
25. As laborer was sliding brick down chute one fell out and hit him on arm. C.
26. Laborer was unloading brick from barrow when brick fell from platform overhead. Cut forehead. D.
27. Brick worked loose from top of wall and hit rigger on head. Cut. C.
28. Laborer was digging ditch with pick when slag flew up, striking knuckle. D.
29. Shoveling ashes, sprained side. D.
30. Lifting concrete slab, sprained back. D.
31. Stepped in hole, sprained leg. D.
32. Stepped on coke, sprained ankle. D.
33. Lifting bag of cement, strained back. D.
34. Stepped on nail, punctured foot. D.
35. Carrying board, lacerated hand. D.
36. Dropped timber, great toe bruised. D.
37. Caught under shaft, fingers cut. D.
38. Dropped casting, bruised toe. D.
39. Rolling timber, contusion of toe. D.
40. Throwing plate on pile, toe contused. D.
41. Putting plate on ditch, toe fractured. D.
42. Rail fell off truck, instep crushed. C.

43. Timber slipped from hand, toe crushed. B.
44. Angle iron dropped, fractured both great toes. B.
45. Picking down clay, cave-in. Bruised chest, back, and hip. D.
46. Catching brick at bottom of chute. Compound fracture of middle finger. B.
47. Handle caught on side of doorway. Sprained wrist. D.
48. Barrow upset and fell from runway with man. Head cut. C.
49. Barrow upset. Index finger crushed. C.
50. Foot slipped, causing fall and upset of barrow. Fractured bones in two fingers. C.
51. Holding barrow on wagon, was thrown onto ground. Leg bruised. C.
52. Crowbar slipped from shoulder. Bruised instep. D.
53. Holding bar to cut concrete, dropped it on toes. Contusions. D.
54. Moving girder with bar and let bar fall on foot. Loss of nail on great toe. D.
55. Moving hoist engine with jack, jack slipped and handle cut forehead. D.
56. Cutting handleleather. knife slipped. Laceration of inside of thigh. D.
57. Hit with sledge while holding bar. Contused instep. C.
58. Cutting peg with hatchet and cut fingers. B.
59. Steam from leaky throttle condensed in exhaust pipe of hoist engine and blew out as man stepped past. Scalded right arm. D.
60. Six men stepped into holes filled with hot water. Two D, three C, one B.
61. Two disabilities from heat prostration. Both D.
62. Pulling weeds. Rash on entire body. D.
63. Two cases of infected blister on palm from use of tools. Both D.
64. Wrenched ankle; did not report injury but rubbed liniment on it, raising a blister which became infected; foot and leg swollen. Reported after 9 days. D.
65. Small cut on little finger. Injured did not report injury but treated cut with carbolic acid. Infection and gangrene of little finger. B.

The yard injuries caused by "Playing," "Falls," "Falling and flying objects," "Hand labor," "Hand tools," and "Infection," have been reviewed as briefly as possible. These injuries are typical of accidents in which the personal factor plays a predominating part. Some unpreventable accidents are a hazard of the work, and are to be expected. However, injuries caused by protruding nails, steam and hot water, infection, falling objects, and falls caused by debris in the yard are preventable, as has been demonstrated by many companies. These causes of injury are particularly amenable to correction by safety committees, as contrasted with accidents in connection with rigging, blasting, running hot metal and cinder, or work with electricity, which must be prevented largely by the foreman in charge.

Aside from covering with guards the cranks and connecting rods of hoisting engines, placing guards about any gears, placing the exhaust where no one can be scalded, and bringing power cables, or steam or air pipe overhead to the temporary locations, not much can be done in the way of safeguarding winches and hoist engines. The safety cleat used in connection with niggerheads is a useful de-

vice, as it enables the man to take a turn of the rope and keep control of heavy material being raised or lowered, when not enough turns have been taken about the niggerhead. In the main, however, accidents with this apparatus result from haste, carelessness, or unskillfulness. The work requires experience to develop skill and few not conversant with it are competent to suggest safe methods. Rigger foremen are best fitted to supervise such work, to eliminate dangerous methods of work, and to devise safe methods. Such foremen should realize the peculiarly responsible position they occupy in this regard, and, welcoming any suggestions from safety inspectors or committees, should take it upon themselves to correct dangerous practices.

The danger of using an electric burner has been so repeatedly pointed out by every furnace man that there is little reason for those concerned not knowing the chances of injury. In accident 8, the injured man, a non-English-speaking foreigner, took it upon himself to try the burner during the noon hour.

If a high-voltage line is carried outside of the plant, it is well to provide beneath the line a platform, access to which is through a locked trapdoor. A similar protection may well be provided inside a plant.

There are several types of safety switches that will prevent accident.

Modern furnaces are nearly all equipped with bleeder valves which either totally prevent any material from being thrown out, or permit only the gas and fine dust to escape, thus partly relieving the pressure generated at a slip. Many furnaces constructed prior to 1902, and some built since, still have two to six easily opening explosion doors which permit red-hot coke, limestone, and ore lumps to be thrown hundreds of yards when a heavy slip takes place. Furnaces thus equipped should have signals provided by which men may be warned of an impending slip. When an explosion door faces a pathway, road, or plant entrance in nearly constant use, it is sometimes feasible to fasten the explosion door shut with a safety bolt of such size that the door will open only on excessively violent explosive slips, where relief is required to avoid blowing off the top or splitting the shell. Some plants provide shields in front of explosion doors facing frequently traveled parts of the yard, or fronting on trestles where the men may be caught inside cars if a sudden slip occurs.

TRACKS.

1. Ball of sand bucket fell on laborer's back. D.
2. As bucket was lowered into car, it bumped end door, causing it to fall on laborer's foot. D.
3. Foreman in stepping from car to crane, had foot on drawhead just as crane backed. Crushed foot. D.

4. Laborer in holding crane hook under car coupling had finger pinched when hook was pulled. C.
5. Laborer had hand pulled into sheave-hook block. C.
6. Laborer was caught between crane bucket and side of car. C.
7. Crane fireman had foot caught between boiler bed and truck frame of locomotive crane. Was getting on just as the crane swung around. Foot crushed. B.
8. Ladle helper tripped on rail. D.
9. Cinder snapper caught heel in guard rail. C.
10. Dust-catcher man stepped on rail and slipped. C.
11. Laborer was on top of coke car getting fuel, when engine bumped car, throwing man to ground. C.
12. Laborer standing on top sill of car tossing brick fell. C.
13. Laborer stumbled over piece of coke. D.
14. Laborer slipped off icy plank. D.
15. Laborer, pushing loaded track truck, slipped on loose coke and fell. C.
16. Ladder broke, allowing rigger to fall from cast-house wall to track. D.
17. Scaffold broke, allowing laborer to fall into empty car, crushing chest. A.
18. Brakeman was struck by pig iron falling from end of car. D.
19. Laborer was struck by cinder falling from passing ladle. D.
20. Laborer was struck on foot by limestone rolling down from stock pile while being loaded with crane. D.
21. Laborer unloading lumber had plank fall on foot. B.
22. Trackman was straightening spike on rail, when spike glanced. D.
23. Laborer was removing ladder from scaffold, when hatchet left on scaffold fell on foot. C.
24. As laborer was unloading ties, helper allowed his end to fall. Tie struck laborer on thigh. D.
25. As handy man was unloading pipe from car, a pipe fell on his shoulder. B.
26. Laborer was using pinch bar to move car when bar slipped, causing him to fall. D.
27. Laborer was opening drop door on car of slag. Door flew open suddenly, causing car wrench to fly about and knock hand against car frame. C.
28. Laborer was pulling spike from track, when spike puller slipped and he fell. D.
29. Track laborer hit foot with sledge. C.
30. Laborer hit fingers with hammer. D.
31. Three laborers were burned by fine dust in shoveling it or unloading it. All D.
32. Laborer was burned by cinder splashing from passing ladle. C.
33. Track laborer stepped on crusted cinder from cracked ladle and broke through. C.
34. Cinder snapper ran water on hot cinder that was escaping from defective ladle; water struck iron and exploded, burning face. B.
35. As laborer was standing near ladle at cast, iron splashed on body. D.
36. Brakeman was uncoupling ladles at hot-metal scales and was at the coupling bar, when hot metal splashed over side of ladle. C.

Familiar dangerous practices of working about buckets without lowering or securing the bail, of standing near end doors, of hooking on, and of grasping cables near the sleeve block are repeated. To prevent accidents like No. 7, a clearance of 18 inches should be pro-

vided between the frames. In 1914 two men were killed by the overturning of locomotive cranes at plants where outriggers and rail clamps were provided but not used. Some cranes are provided with electric indicators to warn crane men when the crane tips to such an angle that further loading will result in overturning. Injuries caused by men being caught between buckets, magnets, etc., and side obstructions while attempting to steady hoisting units before they are let down can be avoided by providing hooks by which the men, working from a safe distance, can steady or stop the swing of equipment.

Stepping on rails in crossing tracks was responsible for five other minor injuries; tripping on rails caused one other injury. Coke along the tracks, insecure ladders, and working without scaffolding are causes that may be eliminated in great part.

When pig iron is loaded, all stray pigs should be removed from the end frames and sides. Ladle-house and cast-house men should remove all large accumulations of scrap from the noses of ladles, and avoid overloading cinder ladles with loose cinder-runner skulls, and should not throw them in so that ends project over the sides. Men should be warned to keep away from cars being loaded or coupled or in transit about the yards.

The danger of handling flue dust and the precautions to be observed have been previously mentioned. Ladles of cinder and metal should not be filled higher than 9 inches from the top, and yardmen should be instructed to keep away from ladles in transit. Brakemen can lessen the hazards of splashes by adjusting couplings of ladles and engines or cars before the coupling is made. The practice of standing close to the sills or frames of road cars to observe whether automatic couplers engage is unsafe when loaded cinder or iron ladles are being coupled. When such ladles are being spotted at scales, at mixers, or pig-casting machines it is usually necessary for the brakeman to stand holding the coupling lever while the ladles are uncoupled; consequently the engineer should use special care not to pull away or to stop abruptly.

Leaks in cinder ladles are nearly always due to cracks, or to iron settling in the bottom and eating through the iron or steel thimble. In attempting to chill leaky cinder ladles with a hose, the workman should remember that iron may be present, and should keep at a safe distance. When it is necessary to venture over cinder in the yard, it is safest to test the surface with a bar or to throw a plank or piece of corrugated iron on the crust to avoid breaking through.

Cracking is the most prolific cause of leaks and of stickers in cast-iron thimbles. Cast-steel thimbles are stated to be relatively immune from cracks.

RAILROADS.

1. Laborer was helping haul coke dust on railroad truck and in walking alongside truck he hurt foot on rail. Bruise. D.

2. Laborer was pushing hand truck with casting on it and wheel ran over foot. Crushed great toe. D.

Foot injuries can be prevented by guards running along the side of the wheels close to the track. This equipment is standard. When heavy material is being unloaded from trucks on tracks the wheels should be blocked because the men have to place or brace their feet close to the rail, and are not unlikely to put their foot on the rail.

3. Laborer was pushing truck loaded with sand bucket; ball fell, crushing middle and third fingers. C.

Bails of buckets should either be put down, provided with a safety catch, or blocked in position.

4. Trackman missed hold and fell in trying to get on moving locomotive. Foot bruised. D.

5. Trackman was closing hopper door, when chain broke, causing man to fall on knee. D.

6. Laborer was tightening brake wheel and caught finger. Laceration of thumb. D.

7. As machinist attempted to jump from moving locomotive crane, foot caught, throwing him and straining his leg. D.

8. Conductor was poling car when pole broke and knocked hand against frame of car. D.

If feasible, it is often safer to pull a car with a cable rather than to push it with a pole.

9. As handyman was climbing over cars he fell and injured knee. C.

10. Locomotive-crane engineer slipped and struck shoulder against throttle of crane, starting engine; foot was drawn into gears. Compound fracture of three toes. B.

11. Brakeman was riding on footboard of locomotive when it derailed, jarring man off and amputated his foot.

12. Brakeman standing on footboard of engine was talking. He carelessly placed his left hand on drawhead when couplers engaged in coupling car. Fingers contused and cut. D.

13. Laborer jumped on footboard of locomotive to ride and was squeezed between car and locomotive as they went about a sharp curve. Bruised hips. D.

14,15,16. Three brakemen injured hands while coupling, injuries varying from thumb bruises, with loss of 3 days, to compound fracture of fingers and burns, in coupling a cinder ladle. B.

17. Conductor was placing car of sand at furnace; car slid when brake was applied, striking concrete wall and catching man's hand between brake wheel and wall. Amputation of finger. B.

18. Laborer was placing his car under crane at cast house but it got beyond his control and bumped into an empty, causing a load of cast-iron plates to skid forward against his feet while he was attempting to set the brake. Left foot crushed. B.

Train crews should not make flying switches. Heavily loaded cars should be spotted by the engine, and should not be uncoupled until they have been brought to a full stop.

19. Cinder man, standing on ladle, was claying cracks when cars broke loose from locomotive putting them on track and bumped the ladle, throwing man to track. Ladle truck passed over arm, crushing same so as to require amputation. B.

20. Millwright was crossing track in front of a string of empties just as the engine bumped them. End car, not being coupled, jumped forward and knocked man down. D.

21. Pig-machine laborer stepped from side of building directly in front of a locomotive running light. Laceration of scalp and bruises of legs, arms, and body. C.

22. Laborer was coking ladles from the yard and stepped backward directly in front of an engine. A.

When a path leading across railroads, exits from doors, or buildings close to railroad tracks presents any hazard of injury from moving cars or locomotives, a railing should be placed so that the approach of men to the track will be retarded sufficiently to compel them to take thought of the possibility of approaching cars (Pl. X, B). When a car is to be switched into a track where men are working on cars or ladles, the men should be warned. Men thus engaged should protect themselves by placing a temporary flag in the tracks back of the switch, so that train men will know that they are in position to be injured. When a locomotive is about to move, or is approaching the ends of buildings adjacent to the track, ends of cars on adjacent tracks, crossings, or any place where workmen are engaged along the track, the bell should be rung. The whistle should be used only for urgent warning. Continuously ringing bells or whistles used for any other purpose than for warning or communication soon come to be disregarded as warning signals. In crossing tracks, men should be instructed to keep at least one car length from the end of a string of cars on the track.

23. Brakeman standing on car struck head against beam as car passed under overhead trestle. Contusion and laceration. C.

24. Brakeman attempted to jump string of coke cars being pushed up yard to furnace trestle, when hand iron pulled off, and man fell under moving train. A.

25. Crane helper was acting as brakeman on ore car which crane was pushing. Fell from the car in unknown manner, and on account of steam, locomotive crane engineer did not see him fall or note his absence until both car trucks had passed over his legs. A.

Few safeguards can be used on railroads about furnace plants. Signs, illuminated at night, showing places of close clearance and telltale ropes at each side of overhead obstructions are essential safeguards, as is a 4-foot fence or railing extending from the cor-

ner of a building and in front of a doorway adjacent to the tracks. Parallel throw switches, guard rails, switch points, bumping blocks at the end of spurs, and automatic couplings on ladles and other track equipment are standard.

Great care is necessary in coupling cars. Couplers are not always uniform in style or size, brakes may be out of order, or cars may move unexpectedly. Couplings should never be made from the inside of a curve. Trainmen should take sufficient time to examine equipment and to adjust couplers, drawheads, and drawbars before exposing themselves to danger in coupling and they should understand that no standing order or custom requires them to endanger life or limb. They should take especial care to familiarize themselves with the situation of overhead and side obstructions.

Jumping on or off of cars, getting in front of moving cars, to adjust knuckles, setting brakes on flying switches, or loosening tightly set brakes are duties attended with danger at all times, and they should not be undertaken when attended with unusual risk. Such work should not be undertaken on any account by members of the furnace force other than trainmen, as the hazard is much greater to unskilled men.

A crossing gate connected with a derailing device is shown in Plate XI, A. When the gate is in a position to protect persons coming toward the crossing the derailing device is thrown off the rail but when it is not in such a position the derailing device is in operating position.

WALKS, PAVEMENTS, SEWERS, AND DITCHES.

1. A hand hook used to regulate overhead gas valve at door of cast house was jarred off by vibration of building, and fell, lacerating scalp of man cleaning up pavement. D.

Hooks on overhead valves should be either securely fastened or taken off when not in use.

2. Stove tender was standing beneath hot-blast valve platform when a bar fell from platform, striking foot. D.

Bars and all loose material should be removed from platforms when work is finished.

3. Rigger stepped on protruding nail outside shanty. Foot punctured. D.

Some one was culpable in leaving such dangerous material about. Everyone should take time to flatten protruding nails or to remove boards or scrap lumber in which nails protrude.

4. Man coming through subway slipped on ice at entrance. Strained shoulder. D.

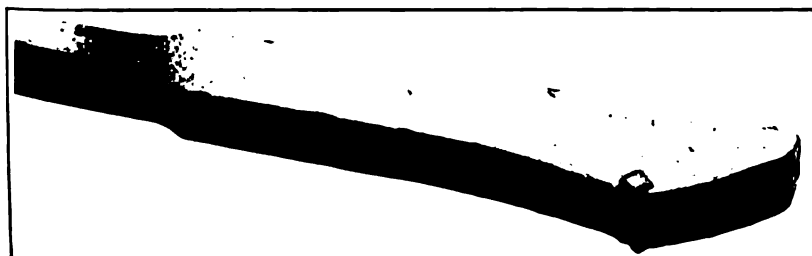
5. Millwright going home fell on icy steps over tracks. Severe bruise and sprained elbow. C.



A. CROSSING GATE WITH DERAILING DEVICE.



B. HANDLING BRICK BY ROLLER CONVEYOR, SAFELY AND EFFICIENTLY.



C. CAR SHIFTER WITH PIECE OF HARDENED STEEL INSERTED IN HEEL.

When one corner of steel becomes worn it can be driven out, turned, and another corner exposed.

In the winter the yard foreman should detail a man to sprinkle salt or ashes on icy walks and steps or to remove ice altogether.

6. Laborer, helping to test cooler plates outside of cast house, let one fall over on his foot. Severe bruises and cuts of toes. B.

The need of especial care in handling heavy castings should be emphasized among members of the labor gang.

7. Rigger's helper was standing in front of exhaust pipe from hoisting engine; engineer started hoist, causing hot water and steam to blow over helper's leg and arm, scalding them. B.

Although the injured man and his fellow workman were each careless, the management was negligent in not having the exhaust discharge guarded or in some inaccessible place.

8. Foreman was pushing scrap down chute from pavement to stock house when he fell down scrap chute into stock house; fractured wrist. B.

Carefulness and the use of safety gloves or hand leathers would lessen the chance of such accidents.

9. Laborer cleaning out hot well was overcome by gas from gas-scrubber discharge, backing up in sewer. Asphyxia and nausea. D.

10. Laborer went into sewer to clean it when explosion occurred which blew hot flue dust over him. Burned face, hands, and wrist. D.

A man working in any sewer or in any well connected to sewers carrying discharge water from gas-cleaning equipment should use every precaution. Half-hour oxygen breathing apparatus should be worn. No open lights or torches should be employed, but guarded electric lights with weather and moisture proof sockets should be used. Ventilation is usually most difficult and stringent precautions, watchers and relief hands should be employed, and the work done under the supervision of a foreman.

11. As laborer was digging in a ditch a board fell and struck him on face and hand. D.

12. Foreman was standing alongside of ditch when it caved in, causing him to fall. Bruised thigh. D.

All loose débris beside ditches should be removed. The sides of ditches and excavations should be systematically tested for firmness and, if loose, should be barred down. Ditches more than 6 feet deep should be cribbed or shored up, and any excavations made in loose earth should usually be cribbed to preclude accidents. The danger is much increased in cold weather when the top becomes frozen. Whenever an excavation is being made near a track, workmen should get out when heavy loads or cars of hot metal or cinders are passing.

13. As teamster was carrying coal across ditch, plank broke, and man fell 2 feet with basket of coal. Sprained wrist. D.

14. Laborer accidentally stepped into sewer manhole from which the cover had been removed. Bruise and abrasion of thigh. D.

15. Engineer stepped into ditch, striking cheek against edge of trench. Fractured bone. D.

16. Laborer was standing on pipe while pumping water out of ditch; he slipped off and cut and bruised right leg. B.

Excavations and ditches should be protected by temporary railings or barriers; lanterns should be placed at night. Plank used for runways or walks should be of standard scaffold lumber, as cribbing or concrete-form boards are apt to be too weak for safety.

Manhole cover plates should never be left off unless the opening is protected by a sign, or portable rail, and by a lantern at night. Whenever it is necessary to work while standing on insecure footing over or near a ditch, a rope should be used for a handhold, or should be fastened about the body, if a fall or slip is liable to result in injury.

HANDLING BRICK AND CEMENT.

1. Laborer missed his footing and fell from brick pile about 18 feet high. Fractured leg. B.

2. Plank runway dropped on foot. D.

3. Brick fell from pile. D.

4. Pile of cement sacks collapsed. Disabilities of 3, 30, 30, and 42 days.

5. Brick pile collapsed; 2, 6, and 15 days.

6. Finger caught while taking brick from chute; 4, 3, 11, 11, and 18 days.

7. Finger caught while tossing brick; 2, 5, and 15 days.

8. Brick dropped in handling caused injuries as follows: To hands and fingers—3, 5, 14, and 24 days; to feet and toes—4, 2, 5, 7, and 14 days; to knee—1 and 9 days.

9. Strains in handling material caused disabilities of 1 and 8 days.

Rules of the United States Steel Corporation regarding the piling of brick and of cement follow:

PILING OF BRICK.

1. Except in brick sheds, brick shall not be piled higher than 7 feet.

2. The pile shall be tied at every course with alternate courses of headers and strikers.

3. When the pile is over 4 feet high it shall be tapered back from a point 4 feet high 1 inch to each foot.

4. In unpling the taper shall be maintained.

5. Under no circumstances shall brick be piled for storage purposes on scaffolds or runways.

6. Tie strips of wood shall be inserted whenever necessary.

PILING OF CEMENT.

1. Cement shall not be piled more than 10 bags high, except in storage bins built for the purpose.

2. The first four end bags shall be crosstied in two separate tiers up to the fifth bag, where a step back of one bag on every fifth bag shall be made. Beginning with the fifth bag, only one cross tier will be necessary.

3. The back tier when not resting against a wall of sufficient strength shall be stepped back one bag to every five bags, the same as the end tiers.

4. Cement bags in outer tiers shall in all cases be piled with the mouth facing the center of the pile.

5. When cement is removed from a pile the length of the pile shall be kept at even height, and necessary step back, every five bags shall be taken care of.

Compliance with these standards undoubtedly would prevent a number of serious injuries. The use of roller tables or conveyors (Pl. XI, B) instead of chutes for brick, together with some instruction in safe ways of taking the brick from the tables, should appreciably decrease the injuries from pinched fingers. The practice of tossing brick is to be discouraged when other means of moving them are available, as injuries are bound to occur. Injuries from dropping brick on the hands and feet result from lack of skill or care, or from the hurry of the work and the fatigue of the men.

MISCELLANEOUS ACCIDENTS AROUND BLAST FURNACE.

1. Laborer was cleaning off top of gas washer and was overcome with gas. Asphyxia. D.

Men should work in pairs at the top of gas scrubbers and there should be steps to the top. If ladders only are provided, they should be fitted with safety cages. Wearing of breathing apparatus is advisable in all gaseous places.

2. Laborer was hit on head by grab bucket when cable broke. Concussion of brain and fractured skull. C.

If possible to avoid it, men should not get under buckets. However, they must sometimes do so, and for that reason especial attention to inspection of cable should be mandatory. The wear of cables is indicated by the breaking of the strands which, as a rule, shows on the surface. Interior wear can be determined by putting two clamps on the rope, a short distance apart, and untwisting the rope slightly, when the inside strands can be inspected.

As the factor of safety of steel cables running over sheaves is usually 5 to 6, breaking of the wires is not serious until 30 to 40 per cent of them are broken within the length of the pitch of the strand. When a cable with 6 strands of 19 wires to the strand has 35 to 45 broken wires in a length in which the strand makes a complete whirl, the cable should be renewed.

End hitches should be made about thimbles with three cable clamps. To avoid slipping of cable through the clamps as the load stretches and thins the cable, the clamps should be inspected and tightened if necessary.

Wear of cables usually results from crystallization of the wire, caused by high speed, small sheave wheels, or long use; from internal abrasion and wear, caused by insufficient lubrication; and from external wear incident to use, or to rough sheaves or sheaves that are too large or too small. The best efficiency is shown when the wires do not commence to break until they are worn nearly half through.

Two cables are used, as a rule, on elevators and skip hoists, and they do not always wear evenly. If one starts to crack the most satisfactory results are obtained if both cables are changed because both will stretch evenly.

Splicing of cables for blast-furnace equipment is rarely satisfactory, as it has to be done by experts to be durable and safe.

On skip inclines there is as a rule less danger from the breaking of cables than from derangement of the skip hoist. Three safety devices which may be said to be essential are an overspeed governor, an automatic stop, and a slack-cable device.

Overspeed devices are possibly more in order on electric than on steam hoists. They are set to act when the speed of the car exceeds its normal rate; for a skip that is operated at a speed of 200 feet per minute, the speed governor is set to throw the current off and apply the brake when the speed reaches 400 to 600 feet per minute.

Automatic skip stops to prevent overrunning at top and bottom are used at practically all furnace plants. Such a stop usually consists of a revolving screw fitted with a traveling nut which engages a pinion as the skip reaches the top or the bottom. This pinion, by an arm or suitable gearing, operates relay switches or the throttle and stops the skip at some predetermined point. The automatic stops are much to be preferred to having the control exclusively in the hands of the skip operator. If adjustment is made to allow for stretching of cable, the hitting of bumping blocks and dangerous and excessive acceleration or retardation of the car are avoided.

Slack-cable devices are employed on electric hoists. Beneath the drum is placed an iron strip held by an arm which is depressed when the cable becomes slack and bears down on it. Depression of the strip causes the circuit to be broken and the dynamic brake to be applied. The device is useful in two ways—if the skip hangs at the top, over the hopper, the device will cause the hoist to be stopped before the cable has accumulated an excessive amount of slack; and if the cables have been allowed to become dangerously slack, so that the car hits the bottom block heavily and starts with a jerk as the hoist accelerates and takes up the slack, the device operates to prevent the hoist from running.

A few plants have additional safeguards—for instance, an additional cut-out device at the top of the skip incline, the device being tripped by the travel of the car beyond a certain point, and being

designed to act when the regular automatic overtravel device fails. By means of a second device an electric circuit is completed through the axle and wheels of the skip as it reaches certain predetermined points at the top and the bottom; the closing of the circuit throws on a light at the skip operator's station. This device is used where the skip operator does not command a view of the skip incline, and is an indication of overtravel, approach to bumping blocks, or of hanging or obstruction of down travel away from the dumping position. A third device is a lever resting on a skip rail and pivoted so that the front skip wheel in coming to the dumping position raises the lever and operates a telltale weight in the stock house. This device is used primarily to indicate that the skip car does not hang at the top.

3. Electrical repair man was standing at foot walk of crane with foot under line shaft, and as shaft started to turn coupling caught roll turned up on bottom of trouser leg and pulled leg under shaft. Compound fracture of right ankle.

Couplings in line shafts should be entirely inclosed with guards that do not revolve with the shafts. Any projecting set screws should either be replaced with safety set screws or be inclosed in a guard.

4. Laborer making a coke fire threw a can of oil onto the fire. Oil burst into flame, burning hands, neck, and face. D.

Men should be emphatically warned against throwing oil on fires. As the use of common hand torches is eliminated, so that oil is less easily obtainable, the liability of this careless act diminishes.

5. Workman was working on the cast-house roof when the furnace slipped and a piece of limestone struck his head, cutting scalp. D.

With old types of furnaces the possibility of such an accident is not remote. Modern tight-top furnaces if equipped with positive closing bleeder valves remain shut against any slip and provide absolute security. Relief valves on bleeders of the free-opening type do not provide any security whatever, as scrap, limestone, and ore lumps are ejected freely through them. Explosion doors on the gas offtakes are exceptionally dangerous.

Whenever men are on top, on roofs, on poles, or in similar places from which escape is difficult, especial precaution should be taken. If the furnace is working stiffly or irregularly, and is slipping, the foreman should not allow men to go to such places, as there may be a slip even when the furnace is not known to be hanging. If the furnace is working regularly, and men are sent to such places, the furnace foreman should be notified, and especial attention should be paid to rodding the furnace. If the furnace hangs or sticks for an abnormal interval, the men should be at once warned to get under cover. If it is necessary that the men go in some place of hazard, the blower should take all necessary precautions. The blast should be taken off if necessary. If the blast is not taken off, the furnace

should be checked. Gangs working in the yard or on the trestle at such times should station a man familiar with the furnace to warn them. Under no circumstances should a man be sent to the top or to a place of similar hazard when the furnace is known to be hanging.

6. Clean-up man was changing electric light, holding wires at base of socket in one hand. The wires crossed, causing a short circuit. Burns of hand and of both eyes. D.

7. Craneman, through with work, was lying on bench reading. Had light wires across leg. Insulation was worn off and wires short-circuited, burning leg. C.

8. Craneman threw in switch and started controller to try out hoist. Threw in controller too fast, and 200-ampere fuse blew out. To escape from flame man leaned out of cab window and fell to ground. Compound fracture of elbow. B.

Fuses on all circuits should be inclosed in an asbestos-lined fuse box, or an automatic circuit breaker should be placed overhead, where the flash will not endanger the eyes. Another alternative is to place the fuse of circuits for lights, larry cars, motors, and hoists at a sufficient distance from switches or controllers to eliminate danger should they burn out. If there is anything amiss with the circuit, fuses generally blow out when the switch is thrown or the controller operated; consequently, fuses and switches should not be placed on the same base, unless the fuse is inclosed. Wire fuses made of base metal, unless inclosed, are particularly dangerous when blowing out because of the flying of molten metal and the heat of the arc.

Wires from electric-light sockets should be taped far enough back to eliminate danger of short circuits. In general, the use of ordinary insulated wire for hand lines, extension lights, or drill motors is to be discouraged. In so far as possible, the light wires should be placed in conduits up to a given point, and standard packing house light cord should be used up to the extension handle. This handle should be of wood, with weather-proof socket. The wires from the lamp socket should not pass through a metal tube, and there should be no connection between the socket and any metal part of the holder, such as a protection cage. Gloves or tongs should be used in replacing globes. It is to be noted that it is not contact with high-voltage circuits, motor frames, brushes, or terminals that causes most injuries from electricity about furnace plants; fuses, electric-light cords, and globes cause most of such accidents. Switchboards, generators, motors, and transformer equipment are generally installed with due regard for safety. The hazards from fuses, hand lights, and cords are less obvious and consequently should be emphasized.

9. Wireman was placing trolley rail when plank on 8-foot scaffold broke, throwing him to ground. Contusion of body. C.

10. Millwright was standing on a ladder resting on a wet floor; the ladder slipped out from under him. Concussion of head and dislocated elbow. B.

11. Handy man was placing metal sheets on roof. He braced himself by placing his foot against a rod, which shook and caused him to fall 40 feet into scrap car. Skull and leg fractured. B.

12. Pipe fitter, sitting on 20-inch steam pipe screwing plug, slipped and fell 9 feet to ground. Fractured skull. B.

13. Rigger, while putting metal sheets on building, was standing on scaffold supported by block and tackle. In handling a sheet, he placed his back against building and braced his feet on scaffold; scaffold swung away from building and he fell 25 feet. A.

GENERAL REMARKS.

The foregoing review of accidents about furnace plants indicates that falls, together with railroad equipment and asphyxiation, account for most of the fatal injuries.

The standardization of scaffold and ladder equipment is admittedly difficult. An employer is in duty bound to make and enforce strict rules in regard to the construction of scaffolds—who is to build them and when and how they are to be built. The range in height of the falls noted above is 8 to 40 feet, and an injury resulting from a 9-foot fall was more severe than one from a 40-foot fall. It seems as if a standard scaffold should be provided for work at any height more than 8 feet from the ground. If the work is of a temporary character, a boatswain's chair may be used.

When a ladder rests on a smooth concrete or iron-plate floor, some one should always hold the base. For a firm wooden, brick, or dirt floor, a ladder with prongs at the bottom is safe, provided it rests squarely at the top. When long ladders, such as are used in tearing out scaffolds inside furnaces, are employed they should be made of the very best 4-inch by 6-inch pine of straight grain and free from defects; or steel ladders may be used. When ladders are used, as is necessary when furnace lining toward base is torn out, each segment should have a 1-inch hemp rope lashed to its top and the rope should be attached to a top structural column. No more than two men should ever be allowed on such ladders at one time and they should keep at least 20 feet apart. A foreman should always be present to enforce these requirements. Recently three men were working on such a ladder and all were killed when the ladder broke.

Jointed steel ladders with safety cages attached are being used to advantage for this work.

In painting or in pipe fitting, rigger, or millwright work on steam or blast lines, ladders should be employed for access to the working point, where if necessary a scaffold may be made. Many steam and blast lines vibrate considerably, and although every furnace force probably has at least one man who is so sure-footed that he can walk along such a line, the act is dangerous and should not be permitted.

Falling of men from roofs occurs so persistently that the necessity of requiring all men who work on roofs to wear belts with safety lines attached has been demonstrated. In one accident of record two men were killed by falling from a blowing-engine roof. They had not been furnished lines nor ordered to use them. In another accident a man fell from a boiler-house roof and fractured both his wrists. Ropes had been furnished and the foreman had been ordered to have the men use them, but he had not done so. His discharge followed the accident.

Falls of workers ranks fourth as a cause of accidents, accounts for the greatest number of fatalities, and results in the greatest loss of time to the workmen.

TABULATED DATA.

Details of the typical accidents outlined in the foregoing review and of other recorded blast-furnace accidents in Pennsylvania in 1915 are shown in tabular form in Tables 1 to 4 following:

TABLE 1.—Detailed data on blast-furnace accidents in Pennsylvania in 1915.

HAND LABOR.

Cause of accident.	Total number of accidents.	Time disability ended.												Total.
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.		
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	
Dropping material:														
Scrap.....	21	15	4	3	9	1	20	1	24	1	30	1	53	53
Pig iron.....	15	7	4	4	10	2	19			1	31		66	66
Castings and plates.....	17	6	5	6	11	2	16	2	28	1	33			
Brick.....	13	9	4	3	12	1	15							
Lumber.....	7	4	4	1	8	1	21			1	35			
Structural material, pipes, etc.....	12	5	3	4	11	1	20	1	27			1	56	56
Miscellaneous.....	3	1	4	1	12					1	32			
Total or average.....	88	47	4.1	22	10.6	8	18	4	27	4	32.5	3	58.3	58.3
Struck or pinched by material:														
Scrap.....	13	5	4	6	8			1	27	1	33			
Pig iron.....	9	2	3	4	11	3	18							
Castings or plates.....	12	4	5	2	13	3	19			2	35	1	60	60
Brick.....	12	3	6	4	12	2	17	2	24			1	38	38
Lumber.....	7	4	5	2	9	1	21							
Structural material, pipes, etc.....	7	1	3	2	9	2	19			1	32	1	120	120
Miscellaneous.....	8	3	4	4	11			1	28					
Total or average.....	68	22	4.5	24	10.1	11	18.5	4	25.8	4	33.8	3	70	70
Strains in lifting or pulling..	24	8	6	9	10	6	17	1	26					
Defective shoes or sharp-edged material.....	35	19	5	10	10	5	18	1	26					
Bumping.....	14	9	5	2	13	1	18	1	26	1	30			
Contact with hot metal.....	9	4	4	4	12	1	21							
Protruding nails.....	7	2	3	4	9	1	15							
Total or average, hand labor.....	245	111	4.4	75	10.4	33	18.1	11	26.1	9	32.6	6	23.1	23.1

TABLE 1.—Detailed data on blast-furnace accidents, etc.—Continued.

HAND TOOLS.

Cause of accident.	Total number of acci- dents.	Time disability ended.												Fatal.	
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.			
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.		
Struck by sledges, hammers, hatchets, or picks:															
By self.....	13	5	5	5	11	3	18								
By others.....	18	11	6	3	13	3	17			1	20				
Total or average.....	31	16	5.7	8	11.4	6	17.5			1	20				
Struck by hand tools of others.....	22	11	5	7	11	1	18			1	35	1	42	1	
Slipping wrenches.....	9	2	3	1	14			1	25	2	34	3	47		
Slipping tools of others.....	26	11	5	7	10	5	18	1	25			2	51		
Strains in using tools.....	7	3	5	3	9			1	25						
Defective tools.....	8	5	5	1	14					1	35	1	100		
Car wrenches.....	34	12	5	10	11	1	17	4	25			5	46	2	
Buggies or wheelbarrows.....	47	20	5	11	12	5	18	5	24	2	34	4	52		
Total or average, hand tools.....	184	80	5.0	48	11.1	18	17.7	12	24.7	8	32.7	16	51.4	3	

FALLING OR FLYING MATERIAL.

Foreign body hitting eye.....	44	27	4	10	10	3	17			1	30	3	100	(a)
Material falling in or from cars and ladles.....	29	12	5	10	12	2	18	1	28	1	35	2	92	1
Material falling from ore chutes.....	22	11	5	5	10	3	18	3	28					
Material flying from sledgeing, chipping, etc.....	17	7	4	4	12	3	19	1	27			2	40	
Tools dropped from elevations.....	7	3	5			2	19			2	32			
Falling timber.....	7	2	7	3	12					1	35			1
Material falling from roofs or eaves.....	4	1	3	1	12	1	16			1	35			
Sliding or collapsing piles or excavations.....	7	2	5	3	11	1	21					1	65	
Falling brick:														
From linings being torn out or repaired.....	13	6	5			3	19	1	22			1	42	2
From walls, scaffolds, or piles.....	6	4	5	1	10	1	17							
From piles collapsing.....	4			1	11					2	30	1	42	
Miscellaneous.....	3					3	16							
Total or average falling brick.....	26	10	5	2	11	7	17	1	22	2	30	2	42	2
Miscellaneous.....	5	3	6	1	14							1	40	
Total or average falling or flying material.....	168	78	4.6	33	12.0	22	17.8	6	26.8	8	32.3	11	68.5	4

(a) Lost sight of one eye.

TABLE 1.—Detailed data on blast-furnace accidents, etc.—Continued.

FALLS OF WORKERS.

Cause of accident.	Total number of accidents.	Time disability ended.												Fatal.
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.		
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	
Stumbling on ground level.	10	6	5	3	11	1	21							
Slipping on ground level.	26	13	5	7	9	1	19	1	26	1	35	4	68	
Falls into ditches or man-holes.	7	2	6	4	12							1	67	
Falls into or in bins.	8	4	6	1	9	2	21					1	120	
Falls in or through cars.	17	6	5	7	10			1	23	3	32			
Falls from permanent platform walks.	5	2	6					1	27			2	71	
Falls on steps.	4	2	4			2	20							
Falls from ladders.	14	1	4	3	11	2	16			2	35	6	57	
Breaking or tilting of plank or runways.	13	5	5	2	12	2	18	1	18	1	30	2	54	
Stepping, slipping, or jumping from elevation.	8	4	4			1	17			1	30	1	76	
Breaking or defect of scaffold.	14	4	6	1	9	1	16			3	30	4	67	
Roofs.	4							1	27			1	40	
Pipes.	5	1	4							2	33	2	53	
Trestle.	4			2	14	1	20					1	180	
Top of furnace or stove.	3	1	2											
Miscellaneous.	10	2	5	2	11	1	17	2	28	2	30			
Total, or average, falls of workers.	154	53	4.6	32	10.9	14	18.5	7	26.7	15	32.1	25	68.8	

BURNS FROM HOT METAL.

Opening and closing tapping hole.	24	9	5	5	11	3	17	3	26	2	30	2	55	
Operating shutters.	11	5	5	1	13	3	18	2	21					
Sparks or splashes.	10	6	4			3	20			1	30			
Cold scrap, bars, and ladles in iron.	9	4	4	2	11	1	21	1	28			1	40	
Water in trough.	6	1	5	2	13			1	28			2	53	
Stepping into runners.	4	2	6			1	15					1	60	
Cleaning runners or troughs.	3			1	10	1	15			1	30			
Pouring at pig machine.	11	5	3	3	13	1	21	1	22	1	35			
Using hand ladles.	5	1	4	2	10	1	16					1	50	
Splashes from track ladles.	5	3	5					2	28					
Skulling and cleaning ladles.	4	1	5	1	12			1	28	1	35			
Breakouts and bursting tuyères.	5	1	6	1	14							3	46	
Babbitt-metal explosions.	4	3	4							1	30			
Total or average.	101	41	4.4	18	11.8	14	18.0	11	25.6	7	31.6	10	50.4	

CRANES, HOISTS, OR RIGGING.

Hooking on.	15	2	4	4	12	1	20	3	27	3	35	2	46	
Caught in gear, rigging, blocks, etc.	21	9	6	4	12	2	18	1	26	3	31	2	81	
Struck by buckets, magnet, or loads.	15	4	4	2	11	2	19	2	26	2	35	3	64	
Slipping or defective hitches.	12	6	4	3	11	1	21	1	22			1	41	
Breaking or loose crane parts.	9	4	6	1	10	1	20	1	28	2	31			
Falling bucket balls.	5	1	6			2	20	1	28			1	90	
Defective rigging.	3											3	90	
Slips or cages.	7	1	3	2	13	2	21			2	31			
Runway accidents.	3											3	49	
Miscellaneous rigging work.	8	6	6							1	34			1
Total or average.	98	33	5.2	16	11.7	11	19.7	9	26.6	13	33	15	663	1

TABLE 1.—Detailed data on blast-furnace accidents, etc.—Continued.

RAILROADS.

Cause of accident.	Total number of accidents.	Time disability ended.												Fatal.
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.		
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	
Larry cars or scale cars:														
Run into by.....		2	5	2	14	1	20	1	28			3	49	1
Car started while being repaired.....		1	6			1	20					2	75	
Miscellaneous.....		2	7			1	20							
Total or average, larry cars or scale cars.....	15	5	6	2	14	3	20	1	28			4	52	1
Setting or loosening broken brakes.....	8	3	6	2	10					1	30	2	66	
Struck by cars or locomotives.....	7	2	4			1	18							4
Jumping on or off equipment.....	6	2	5	3	13	1	21							
Track trucks.....	5	3	4	1	11			1	25					
Coupling.....	4	1	3	2	11							1	60	
Bumping cars.....	4	1	3	1	13							1	60	1
Falls of persons.....	4	1	5											3
Locomotive crane.....	4	1	17	1	14							2	66	
Close clearance.....	3	1	4			1	17					1	70	
Derailment.....	2	2	5											
Miscellaneous.....	6	1	4	2	12	1	21	1	26					
Total or average.....	67	23	4.9	14	12.9	7	19.6	3	26.3	1	30	11	60.1	9

MACHINES OR MACHINERY.

Engines or pumps:														
Crank, connecting rods, or crossheads.....		2	5	1	9							1	42	
Valve mechanism.....		1	7	1	8	1	16	1	26					
Wrist plates or rocker arms.....		2	4			1	19	1	25					
Miscellaneous.....		1	5	2	14	2	16					1	42	
Total or average.....	18	6	5	4	11	4	19	2	25			2	42	
Machine tools, emery wheels, or steam hammers.....	10	2	3	1	8	4	19	2	27			1	42	
Bin-door mechanism.....	7			4	13	1	21			1	31	1	60	
Caught in belts.....	5	2	5	1	10			1	25			1	60	
Ladle-dumping mechanism.....	3			1	10	2	20							
Clay-gun mechanism.....	3	1	3	1	8			1	25					
Car-door mechanism.....	3	1	17			2	18							
Breaking machinery.....	2	1	3			1	21							
Unclassified.....	6	1	5	3	8	1	21					1	42	
Total or average.....	57	14	4.6	15	10.4	15	19.4	6	25.7	1	31	6	48	

FLAMES OR EXPLOSIVES.

Torches.....	11	1	7	4	13	2	20			2	32	2	76	
Flames from tuyères.....	9	3	7	4	13							2	48	
Explosion of gas at top or at mains.....	9	3	4	2	11	1	15	1	28			1	40	1
Lighting gas.....	7	3	4	4	10									
Oil on fires.....	2			2	14									
Stove doors blown off.....	2									1	35			1
Explosives.....	1	1	9											
Total or average.....	41	11	5.5	16	12.1	3	18.3	1	28	3	73	5	57.6	2

TABLE 1.—Detailed data on blast-furnace accidents, etc.—Continued.

BURNS FROM CINDER.

Cause of accident.	Total number of accidents.	Time disability ended.												Fatal.
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.		
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	
Splashes.....	6	1	4	2	9	1	19					2	98	
Breaking hot cinder.....	4	3	5									1	75	
Splashes from tapping hole or blow pipes.....	5	2	5			1	20			2	32			
Tapping cinder.....	6	2	7	2	10	1	16					1	42	
Botting up.....	4	1	3	1	8	2	16							
Dumping ladles.....	6					1	21			1	35	3	51	1
Stepping into cinder.....	3			1	10					2	30			
Boils.....	3	2	7			1	16							
Transportation.....	2					1	15					1	101	
Clothing ignited from heat of cinder.....	3			1	10	1	21			1	42			
Total or average.....	42	11	5.5	7	9.3	9	17.7			6	33.5	8	70.9	1

ILLNESS.

Infection:														
From cuts, punctures, or slivers.....	13	3	4	4	11	3	19	1	22	1	33	1	56	
Handling tools.....	10	2	6	6	12	2	18							
Bruises.....	6			2	10	2	19	1	24			1	40	
Home treatment.....	3			1	10							2	53	
Others.....	2	1	7							1	28			
Total or average.....	34	6	5.1	13	11.2	7	18.7	2	23	2	30.5	4	50.5	
Overcome by heat.....	4	3	4	1	9									
Fainting or fits.....	2	1	4	1	9									
Total or average.....	40	10	4.7	15	10.8	7	18.7	2	23	2	30.5	4	50.5	

FIGHTING OR PLAYING.

Fighting or playing.....	3	2	2	1	8									
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ELECTRICAL MACHINERY.

Fuses.....	11	4	3	3	13	1	16	1	28	1	30	1	132	
Globes or sockets.....	7	4	5	3	12									
Short-circuited wires.....	6	1	7	2	13	8	20							
Pulling switches.....	4	2	5	2	9									
Tools slipping.....	4	2	3	2	9					1	30			
Contact with electric equipment.....	3	2	4											1
Electric burner.....	2	2	4											
Total or average.....	38	17	4.2	12	11.4	4	19	1	28	2	30	1	132	1

TABLE 1.—Detailed data on blast-furnace accidents, etc.—Continued.

HOT WATER OR STEAM.

Cause of accident.	Total number of accidents.	Time disability ended.												Fatal.
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.		
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	
Stepping or falling into steam or hot water.....	27	6	6	4	12	4	19	2	26			5	58	
Defective or bursting equipment.....	14	4	4	3	12	2	18	1	24	2	32	2	48	
At clay gun.....	6	1	7	3	13	2	17							
Steam from exhausts.....	3			1	13					1	30	1	45	
At cinder pit.....	4	1	7					2	23			1	45	
Water on hot material.....	4	2	5	2	13									
Hot water or steam from turbines.....	2	2	5											
Uninsulated pipe.....	2	1	2			1	21							
Mishandling valve.....	3	1	3	1	9	1	15							
Total or average.....	59	18	5.1	14	12.2	10	18.2	5	24.4	3	31.3	9	81.5	

SLIPS.

Hit by material from slips.....	12	6	5	3	11	2	19					1	70	
Injury received while running away from slip.....	2	1	6											1
Flames from stove or boiler doors at slip.....	2	1	4					1	22					
Total or average.....	16	8	5	3	11	2	19	1	22			1	70	1

ASPHYXIATION.

At boilers.....	11	4	5			2	19	1	25			1	56	3
At gas engines.....	5	4	6			1	16							
At gas washers.....	2	2	4											
At furnace tops.....	2	1	7			1	20							
At inside stove.....	1	1	6											
Total or average.....	21	12	5.4			4	18.5	1	25			1	56	3

HOT FLUE DUST.

Stepping into hot flue dust.....	6			1	11	2	18			2	32	1	38	
Flying or falling hot flue dust.....														
Unloading cars.....	7	1	2	4	10							2	46	
Boilers.....	4	1	7	2	12							1	57	
Gas mains or stoves.....	6	2	4	1	14	1	21	1	28			1	60	
Chutes.....	3			2	12			1	25					
Flue-dust plant.....	3	1	4	1	10	1	16							
Putting water on.....	2			1	14							1	56	
Total or average.....	31	5	4.2	12	11.3	4	18.3	2	26.5	2	32	6	50.5	

TABLE 2.—Data regarding duration of injuries.

Cause of injury.	Percentage of injuries that terminated after second week.	Average time lost from injuries causing disability of more than two weeks.
		<i>Days.</i>
Hand labor.....	24.1	20.9
Hand tools.....	30.4	31.4
Falling or flying material.....	30.9	31.8
Falls of person.....	45.8	51.9
Burns from hot metal.....	42.2	29.0
Cranes, hoists, or rigging.....	47.3	36.3
Railroads.....	44.8	38.9
Machines or machinery.....	49.2	27.3
Flames or explosives.....	34.1	38.5
Burns from cinder.....	57.1	46.4
Illness (including infection).....	37.5	28.3
Electric machinery.....	23.7	42.3
Hot water or steam.....	44.1	31.9
Slips.....	31.2	32.5
Asphyxiation.....	28.5	25.8
Hot flue dust.....	45.1	37.1

TABLE 3.—Summary of data regarding causes of accidents and duration of injuries.

Cause of accident.	Total number.	Time disability ended.														Fetal.
		First week.		Second week.		Third week.		Fourth week.		Fifth week.		Sixth week.				
		Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.	Number.	Days.			
Hand labor.....	245	111	4.4	75	10.4	33	18.1	11	26.1	9	32.8	6	63.1			
Hand tools.....	184	80	5.0	48	11.1	18	17.7	12	24.7	8	32.7	16	51.4			
Falling or flying material.....	168	78	4.6	38	12.0	22	17.8	6	26.8	8	32.3	11	68.5			
Falls of person.....	154	53	4.6	32	10.9	14	18.5	7	26.7	15	32.1	25	68.8			
Hot metal.....	101	41	4.4	18	11.8	14	18.0	11	25.6	7	31.6	10	50.4			
Cranes, hoists, or rigging.....	98	33	5.2	16	11.7	11	19.7	9	26.5	13	32.8	15	66.3			
Railroads.....	68	23	4.9	14	12.9	7	19.6	3	26.3	1	30.0	11	60.1			
Hot water or steam.....	59	18	5.1	14	12.2	10	18.2	5	24.4	3	31.3	9	51.5			
Machines or machinery.....	57	14	4.6	15	10.4	15	19.4	6	25.7	1	31.0	6	48.0			
Flames or explosives.....	41	11	5.5	16	12.1	3	18.3	1	28.0	3	33.0	5	57.6			
Burns from cinder.....	42	11	5.5	7	9.3	9	17.7			6	33.6	8	70.9			
Illness (infection).....	40	10	4.7	15	10.8	7	18.7	2	23.0	2	30.5	4	30.5			
Electric machinery.....	38	17	4.2	12	11.4	4	19.0	1	28.0	2	30.0	1	132.0			
Hot flue dust.....	31	5	4.2	12	11.3	4	18.3	2	26.5	2	32.0	6	50.5			
Asphyxiation.....	21	12	5.4			4	18.5	1	25.0			1	56.0			
Slips.....	16	8	5.0	3	11.0	2	19.0	1	22.0			1	70.0			
Fighting or playing.....	3	2	2.0	1	8.0											
Miscellaneous.....	6	3	6.0	2	11.0			1	28.0							
Total or average.....	1,372	530	4.7	338	11.1	177	18.3	79	25.7	80	32.3	135	60.9	33		

Proportion of nonfatal injuries after second week, per cent..... 35.1

Duration of injuries terminating after second week, days..... 34.1

TABLE 4.—Summary of data regarding location of accidents.

Location of accident.	Total number of accidents.	Cause of accident.															
		Hand labor.	Hand tools.	Falling or flying material.	Falls of person.	Burns from hot metal.	Cranes, hoists, or rigging.	Railroads.	Machines or machinery.	Burns from cinder.	Flames or explosions.	Illness (infection).	Electric machinery.	Hot fine dust.	Asphyxiation.	Slips.	Miscellaneous.
Car dumper or ore bridge.	40	5	2	5	5	...	11	7	1	...	...	4	...	...	...	...	...
Trestle or ore yard.	124	19	38	21	20	...	3	9	4	...	1	5	1	2	...	1	...
Stock house.	117	30	34	30	6	...	...	1	6	...	2	3	1	2	...	1	1
Scale car, skip, or hoist.	37	1	3	4	4	...	4	15	1	...	1	...	3	...	...	...	1
Furnace top or stack.	44	8	7	5	11	...	3	...	...	5	2	3	...	...	2	...	1
Blowpipe, tuyere, bosh, or bustle pipe.	29	...	5	6	2	1	2	...	...	3	8	...	...	...	...	...	2
Tapping hole or cinder notch.	89	3	13	...	4	33	4	...	1	22	2	...	1	...	...	...	6
Runner or bed.	78	26	8	5	3	27	2	...	2	...	...	4	...	...	...	...	1
Elsewhere in cast house.	98	32	11	8	7	8	8	2	...	...	1	6	4	...	...	2	6
Pig machine or ladle house.	110	12	9	18	8	22	15	...	7	2	3	3	3	...	...	...	8
Slag or fine dust disposal.	58	7	4	3	7	...	7	5	5	8	...	2	2	...	8	...	...
Gas-main system.	35	...	...	9	8	...	5	...	1	...	2	1	1	6	2	...	...
Pig iron or other storage.	69	43	3	10	5	...	5	2	...	...	...	...	1	...	...	...	...
Stoves.	39	6	5	8	3	...	2	...	...	...	4	1	1	4	1	2	2
Boilers.	91	11	6	6	14	...	2	...	...	...	3	3	1	6	11	1	17
Engine rooms.	76	4	5	7	9	1	6	...	19	...	4	1	6	4	4	4	2
Shops.	43	3	8	7	2	3	2	...	11	...	1	2	3	...	...	...	1
Yards.	91	21	7	7	13	1	6	7	...	2	...	6	2	3	...	8	8
Tracks.	68	6	7	5	13	5	8	20	...	3	...	...	1	...	...	...	...
Pavements or sewers.	21	2	2	4	7	...	2	...	...	...	1	1	...	...	1	1	...
Miscellaneous.	16	6	...	...	3	...	1	...	...	...	3	...	3	...	...	...	...
Total.	1,372	245	184	168	154	101	98	68	57	42	41	40	38	31	21	16	59

PLACING OF RESPONSIBILITY FOR ACCIDENTS.

DIFFICULTIES INVOLVED.

The classification of accidents according to responsibility is unsatisfactory. An absolutely correct analysis of the many factors and circumstances involved is impossible, because complete details of most accidents are lacking and because the placing of responsibility must largely depend upon the decision of the compiler as to assumptions as to fact and as to other details. An example may illustrate the difficulties. In one plant a man fell from the cast-house roof. He may have been provided with belt and line and ordered to use them, but because of some feeling of bravado may have neglected to do so. Conversely, no safeguard may have been provided. Assuming that safeguards were available, the question remains as to whether the foreman gave the employee sufficient and mandatory instructions, and, finally, the question remains as to the length to which foremen and superintendents must go. Can a foreman relieve himself of responsibility for injury to his men by giving them clear and definite instructions and by providing safeguards, or is he under obligation to follow each man to his place of work and personally see that the man continually uses the safeguards or does his work according to instructions? The cause of this accident could possibly be placed under any one of five different heads, as follows: Lack of guards, nonuse of guards, insufficient instructions, lack of supervision, or acting against rule; or, broadly, it might be classified under one of two heads—fault of workman or fault of employer.

At another plant a man was overcome with gas while changing a gas burner. The accident might be attributed to carelessness or want of skill in staying in the gas too long, to acting against a rule, to nonuse of safeguards—that is, breathing apparatus—to lack of safeguards (breathing apparatus), or to deficient plant equipment, such as lack of valve with which to turn off gas from the burner. At several plants men were severely injured in opening car doors with car wrenches. Such an accident might be classified under lack of guards, lack of skill, or hazard of the work. Lack of guards implies that some type of safety car wrench should have been employed, but similar accidents have occurred even with safety wrenches. Lack of skill implies that the employee did not have the ability to know by inspection the load on the winding stem and the car door, which is

an unreasonable requirement, but employers have heretofore claimed that such accidents should be attributed to lack of skill.

Another example is injuries that are caused by explosions of pockets of hot metal when the iron trough is chilled with water after a cast. Hazard of the work, carelessness, acting against rule, nonuse of safeguards, lack of safeguards, insufficient instructions, and lack of supervision are possible classifications for such injuries.

CLASSIFICATION OF BLAST-FURNACE ACCIDENTS ACCORDING TO RESPONSIBILITY.

A classification of the causes of accidents that has proved sufficiently comprehensive and is believed to be reasonably accurate follows:

Classification of blast-furnace accidents showing assignment of responsibility.

Cause of accidents.	Per cent.
1. Hazard of work and inevitable risk.....	29.8
2. Carelessness or lack of skill.....	14.2
3. Acting against rules.....	3.5
4. Nonuse of safeguards provided.....	1.4
5. Lack of safeguards.....	5.6
6. Insufficient instructions.....	2.1
7. Insufficient supervision.....	2.1
8. Deficient plant arrangement and equipment.....	7.0
9. Fault of fellow workman.....	2.7
10. Miscellaneous and accidents whose cause was not disclosed....	31.6

The percentages are given with the reservation that the large number unclassified, if they could be properly placed, would change some of the totals materially. Most of the unclassified accidents would fall either in class 1 or class 2. It is not considered logical, on the basis of indefinite data, arbitrarily to charge the industry with a great hazard nor the workman with an undue lack of skill or care. The remaining totals would be little changed. If similar causes be grouped, the classification would be as follows:

Hazard of industry.....	29.8
Fault of workman.....	21.8
Fault of employer.....	16.8
Miscellaneous.....	31.6
	100.0

This classification differs radically from classifications in which 45 to 85 per cent of the injuries are attributed to the fault of the workman. In regard to "hazard of the industry," or, in other words, the inevitable risk, it may be stated that this term is elastic, and its application to accidents at different plants shows the present state of plant conditions, equipment, and efforts to prevent accidents as the term is regarded differently at different plants. Many hand-

labor and hand-tool accidents may be assigned to this class instead of to lack of skill or carelessness. The handling of material or the use of hand tools calls for the assumption of more or less risk, owing to the fact that men can not attain a machinelike precision. At the same time most of such accidents are directly caused by clumsiness and lack of skill, and consequently their number can be lessened, and some of them are, therefore, not to be attributed to inevitable risk.

"Lack of skill" and "carelessness" are grouped together because distinction would be difficult as regards accidents not witnessed or described fully. Under this heading are necessarily included such causes as ignorance and imprudence, inefficiency and inattention, and praiseworthy and foolhardy efforts.

"Acting against rules," "insufficient instructions," "nonuse of guards," and "insufficient supervision" are closely related. "Rules" about a furnace plant could be multiplied to fill a volume. However, they largely cover certain well-understood practices, departure from which invites accident. Infraction of rules is relatively rare, and is responsible for few accidents. Accident-prevention efforts have introduced new rules, novel to a considerable proportion of the force, relating to the use of goggles, safety belts and lines, condition of tools, construction and use of scaffolds, ladders, and rigging tackle, use of safeguards, handling of electrical appliances, etc. Some of the rules and safeguards undoubtedly have been formulated and designed more as a result of enthusiasm than of specific knowledge. Some have not been adapted to actual furnace conditions, and have interfered with production, and the men, therefore, have not complied with them, nor have the foremen required compliance. A feeling that little work worth while can be done without exposure to danger, and familiarity with and contempt for danger has led to disregard of other rules, and resulting accidents have been placed under "acting against rules." To what extent this cause is augmented by the tacit acquiescence of foremen is problematical, but there are sufficient data to indicate that it is an appreciable factor. Accidents in which that factor appears are included under "insufficient instructions" and "insufficient supervision," which include accidents in which men are palpably ignorant of obscure hazards or are assigned to especially hazardous jobs without the supervision or assistance of a foreman, subforeman, or capable helper. Most men resent continued supervision, suggestions, and instructions; they prefer to work in their own way. Further, it is literally impossible for a foreman to personally supervise every man's work. Also, it may be questioned whether initiative and resourcefulness are not stifled when supervision is extended too widely. The other extreme is represented when an injury occurs from lack of supervision or instructions, and such

injuries are sufficiently in evidence to indicate that instruction of men and supervision of unusual or extrahazardous work should be increased in some plants.

"Lack of safeguards" and "deficient plant arrangement and equipment" together cause 12.6 per cent of blast-furnace accidents. In regard to safeguards few furnace plant managements fail to use safeguards on account of their cost. However, at several plants there is a pronounced disposition to reject some safeguards in the belief that their use is unreasonable, that they are impracticable or inefficient, that they will interfere with the work, or that the risks involved are only seeming, not real. In the figures given, "lack of safeguards" covers accidents that would be prevented if guards in use at some plants were used. "Deficient plant arrangement and equipment" covers many accidents that might be charged to the inevitable risk of work at some plants. Practically every plant has some characteristic handicap in layout, design, or equipment to which is ascribed deficiencies in output, fuel economy, or "cost above." To corresponding plant conditions a certain percentage of accidents is due, and although such accidents are not, strictly speaking, the fault of the employer, the responsibility for the condition is his. Such a condition is, as a rule, a heritage from former days and tends to be eliminated by the demand for operating economies.

CHARACTER AND VALUE OF PRINCIPAL BLAST-FURNACE SAFEGUARDS.

The efficiency of safeguards in preventing accidents about a blast-furnace plant is not questioned, so far as is known, by operating men. However, the use of a guard often involves extra work in placing and removal, or in handling during a job, with no resultant gain in production. In contrast are certain safeguarding appliances employed in steel mills, or for wire-drawing benches and many kinds of machines. The device so used is a permanent and integral part of the machine itself and frequently results in an augmented production or efficiency.

Although many safeguards may not pay in direct financial return by preventing all of the infrequent accidents for the prevention of which they are designed, it is established that most safeguards do pay when supplemented by other accident-prevention methods. The safeguards furnish evidence of the earnestness of the company to lessen the hazards and emphasize its efforts to procure the cooperation of the force in reducing accidents. Thus it can be said that the installation of a safeguard is rarely lost effort.

A list of safeguards employed at various plants follows:

List of blast-furnace safeguards.

CAR-DUMPER SAFEGUARDS.

Lights at approach and discharge.
 Fenders on truck wheels.
 Inclosed cradle counterweights.
 Steps to operator and motor house.
 Electric trolley or "third rail" elevated 8 to 12 feet above yard level or inclosed in underground conductors.
 Overtravel device for cradle.
 Permanent railed platform about hopper of stationary car dumper.
 Gong to notify operator when men are off from cradle.

SAFEGUARDS FOR LARRY OR TRANSFER CAR.

Fenders of the trolley-car type.
 Air whistle or hand horn.
 Headlights.
 Air brakes.

ORE-BRIDGE SAFEGUARDS.

Truck-wheel fenders and guards.
 Inclosed gears on drive and transmission.
 Caged ladders up gantry legs.
 Fireproof operator's cage.
 Steps to platform of operator's cage.
 Floor of cage extended to form landing platform and railing around platform.
 Automatic warning bells.
 Bumpers at ends of rail girders.
 Rail clamps beneath trucks.
 Footwalk with double railing entire length of bridge and across ends.
 Main-line switch outside of cage and at each end of bridge at the trucks.
 Trolley or third rail elevated or placed outside trestle work.

TRESTLE SAFEGUARDS.

Railed walks with outside edge 6 feet from rail and inside edge directly beneath car frame.
 Crossovers from each bin to enable men to get to opposite side without jumping.
 Gratings, cable, and pipe over bins between track girders. Also similar cover between walk and track girders.
 Use of safety belt and line in entering bins and in unloading cars.
 Use of pinch bars, each with disk on handle and tool-steel insert in heel.
 Use of safety wrenches of various types.
 Use of magnet to unload heavy scrap.

STOCK-HOUSE SAFEGUARDS.

Platform for barring ore from chutes.
 Chutes provided with auxiliary heavy sluice gates to cut off flow of material.
 Arrangement of levers and cables for operating chutes so that operator is out of reach of falling or flying stock.

Bin door or chute mechanism guarded and provided with oiling platform.
 Quick-stopping device for driving shafts.
 Fenders, air brakes, and gongs or air whistles for scale cars.
 Overhead trolley for scale cars.
 Ample clearance for men on side of scale car or recesses in walls at intervals.
 Railing between scale-car track and entrance to stock house.
 Passageway from beneath chutes and outside of tracks.
 Warning sign placed at empty bins about to be filled.
 Knuckle guards on barrow handles.
 Arrestor rail about skip pit to prevent hand barrows from falling into pit.
 Steps or ladder to bottom of skip pit.
 Clearance about skip car at sides and bottom of pit.

HOIST-HOUSE AND SKIP-INCLINE SAFEGUARDS.

Locked doors at entrance.
 Inclosing entire drum with removable wire-netting guards.
 Railings about drum and motor or engine.
 Hoisting cable run through chute to roof opening.
 Connecting rods and cranks guarded.
 Gears and screws on overtravel devices guarded.
 Overspeed and slack-cable devices on hoist motors.
 Locks for throttles and switches of hoist engine and motors.
 Standard guarded switch boards.
 Skip track equipped with shield at point 7 to 10 feet from stock house and yard level to prevent men from putting hands on rail.
 Skip track equipped with shield on underside from top of furnace.
 Overtravel device at top of skip tracks.
 Safety clamp on cages thrown on by spring.
 Sides of cage inclosed with sheet iron or small-mesh woven wire.
 Cover over top of cage.
 Gates at both top and bottom landings automatically operated by travel of cage.
 Providing top platform with safety catches that secure cage at top.
 Platform at sheave wheels.

SAFEGUARDS IN CONNECTION WITH INDICATING DEVICES.

Signal to cast house or blower's office to inform of furnace hanging.
 Speaking tube or telephone to furnace top.
 Locks for bell and skip operating levers.

SAFEGUARDS FOR FURNACE TOP.

Shanty for top fillers, located behind or at side of hoist, with window opening away from furnace.
 Rail about hopper to catch barrows.
 Railed platform at top of receiving hopper.
 Guards about distributor mechanism.
 Permanent steps or caged ladders to sheave-wheel and other platforms.
 More than one means of approach to furnace top.
 Means of plugging rodding hole when working on top.
 Stretcher to lower men from top where descent is difficult.
 Telephone, speaking tube, or electric-gong signal from top to stock house or cast house.
 Steam connection to gas seal over big bell.

Platforms connecting explosion doors, bleeder valves, and outriggers.

Bolting or chaining a number of explosion doors to limit their opening, or placing shields in front of doors.

Use of bleeder valves or explosion valves that allow only fine ore and gas to escape.

Elimination of "bracket and plate" top construction.

Elimination of rodding of furnace from top.

House or shelter on top to protect men from slips.

FURNACE-STACK SAFEGUARDS.

Railed platform about shell to facilitate use of sprays for hot spots.

Shells double and triple riveted and with butt straps and of thickness up to 1½ inches.

Corrugated-iron or steel wearing plates in place of, or in front of, brickwork at stock line, eliminating repairs during blast.

Permanent railed platforms about stack to allow access to cooling plates.

Steps or caged ladders to stack platforms equipped with a warning device to be operated when steps or ladders are in use.

Railed platforms on bustle pipes.

Bosh bands 1½ inches thick.

Bosh plates anchored in boxes.

Tuyère breast armored with 1-inch steel plate or cast-steel jacket.

Strong bridle posts, rods, and springs.

Auxiliary bridle rods.

Elimination of pivoted counterweighted eyesights.

Elimination of spray cooling of hearth jacket.

Elimination of ditch about hearth jacket.

Hearth jackets to extend 4½ to 5 feet below iron notch.

Strong hearth jacket up to 6 inches thick, and strong bands up to 2 inches thick.

Adequate tying in of hearth-bottom brickwork.

Two cinder notches.

Water-deflecting shields over skimmer trough and cinder-notch runner hung from bustle pipe.

CAST-HOUSE SAFEGUARDS.

Railed sides and railed or grated casting holes.

Runways at exits of cast houses.

Steel plate ¼ to ½ inch thick on roofs with steep slope.

Ventilator about stack.

Short runners and railings along runners.

Bridges over runners.

Shutters operated from a safe distance by cables, steam cylinders, or counterweights.

Shields at hand-operated shutters.

Footpieces on shutters.

Shields over or below skimmer to prevent coke and slag from blowing down into cast house.

Shields at punch out and monkey skimmer.

Raising of punch-out gate and splasher from safe distance by cables.

Long pipe for iron-chilling trough.

Electric drills for opening tapping holes.

Shields for use in drilling or driving bars at tapping holes.
 Automatic mud guns.
 Funnels on clay cylinder and shield on nose of hand-operated guns.
 Steam exhaust turned up and away from rear of gun.
 Automatic cinder-notch bots.
 Adjustable shields for hand botting.
 Checkered floor plates.
 Auxiliary signal to blowing room.
 Protection plate in front of snort-valve lever.
 Relay system by which signal to blowing room sounds "buzzer" at skip and bell operator's station.
 System of red and white electric-light signals by which the hanging and slipping of the furnace can be signaled from stock house to cast house, or reverse.
 Signal to boiler house from cast house.
 Signal to gas washer from cast house.
 Warning whistle, lights, or signs for slips, operated from cast house.
 Catch basin for iron in runners leading to ladles and granulating pit.
 Oxygen burner for hard or ironed tapping holes.
 Means of holding "welshman" when bar is backed from tapping pole, so that no one is exposed to burns from flying iron.
 Means of holding bar being driven into tapping hole to keep men out of range of misdirected stroke of sledge.
 Water-supply pipe for cinder-notch coolers extending one-third or one-half around furnace before rising to circle water-supply pipe.
 Elimination of cooling plates or boxes about sides of tapping hole.
 Steep slope of skimmer iron trough.
 Goggles, masks, leggings, box-toe shoes, and safety or gauntlet gloves provided at cost.

SAFEGUARDS IN CONNECTION WITH IRON DISPOSAL.

Heavy bottoms in iron ladles and ladles equipped with devices for keeping them upright on frames or cars. Automatic couplers and safety chains. Steel underframes between trucks.
 Each dumping crane or winch equipped with cable attached to hook fitted with hand hold, instead of chain and common hook.
 Pourer at pig machine protected from splashes.
 Trough men furnished with masks or goggles, and shields placed at end of spout over mold, or shields behind which men may work.
 Walks over strands and runners and between strands from front to back.
 Automatic knockers at discharge end of mold.
 Shields of wire mesh at end of chain, also along car track opposite chutes.
 Railings along columns of pig machine, except at passageway, which should be covered with plate shield.
 Steam valves for lime sprags placed out of path of molds.
 Lime run into lime vats by spouts from mixing tanks.
 Standard guards for engine or motor, guns, chains, and transmission equipment.
 Quenching pits with heavy brick walls or steel-plate shields, behind which men work.
 Iron chills or depressions in yard into which ladle cleanings are dumped, and shields behind which men can work.
 Ladle pits in floor of ladle house, provided with coping walls.
 Part of ladle house in which skulls or scrap is broken to be inclosed for protection of man dropping ball.

GUARDS FOR GAS-MAIN SYSTEM.

Elimination of counterweighted explosion doors on downcomer, dust catcher, and mains.

Inclosure for base of dust catcher.

Operating device for bell lever on dust catcher that permits dumping by man standing a safe distance away.

Clamp to hold dust-catcher bell lever tight in shut position.

Provision of large (4-inch) sprinklers to wet dust thoroughly as it is dumped into car.

Steam inlet to dust catchers.

Water seals in conjunction with valve mechanism at gas main offtake from dust catcher to prevent explosions or asphyxia.

Bleeder device at dead end of gas main.

Downlegs from gas main brought low to prevent flying of dust.

Clamping of bells on downlegs, and cables to open bells from safe distance.

Manholes of adequate size.

Close spacing of manholes.

Railings and platforms along tops of gas mains, with permanent ladder approaches.

Water-seal valves at points where parts of the system are to be isolated, with clamps for the bells.

Frequent downlegs, V pockets, or washing facilities, requiring less frequent work inside mains.

Pressure gage, reading in inches of water, to indicate pressure of gas in mains.

SAFEGUARDS AT GAS WASHERS.

Steps to top and to cleaning doors, side sprays, or water inlets.

Water-seal and mechanical valves on the inlet side, and water-seal valves on the discharge side.

Adequate ventilation by fans or exhausters when men are cleaning inside.

Rotary washer outside of building.

Fans for ventilating scrubber or gas-cleaning buildings.

System for controlling from gas house gas sent to engines and stoves.

Signals to boiler and engine operators.

Means of stopping ignition of engine in emergency.

Hourly registration by operator of cleaning house or inspection by watchman to determine operator's safety.

Automatic signal to warn operator of low-gas pressure.

BOILER-HOUSE SAFEGUARDS.

Provision for securely stopping gas from leakage past valve through burner into boiler when shut off for cleaning.

Blanking or locking of boiler blow-off valves.

Gage-glass shields that can be rotated when the glass is being changed.

Inward-opening firing doors.

Escape pipes from pop safety valves.

Best wire-wound steam hose for blowing tubes.

Nonreturn valves on steam nozzle.

Gas mains placed outside boiler house.

Permanent railed platforms from boiler to boiler, and for water tender, or permanent ladders for use when gage glasses are changed.

Permanent platforms with ladder or other approach at frequently used valves.

Permanent platforms for tube blowers.

SAFEGUARDS FOR STOVES.

Platforms at cleaning doors provided with means of approach that obviates climbing over top of railings.

Steps to top provided with frequent landings, with danger sign at bottom.

Locks for gas burner, hot-blast valve, cold-blast valve, and chimney valve.

Provision for preventing leakage of gas from burner into stove.

Means of turning gas into stove without man standing too close to door.

Blow-off valve muffled or provided with a 16-foot vertical extension pipe.

Hot-blast valve counterweights guarded.

Permanent platforms at hot-blast valve seat and valve chamber head.

BLAST-MAIN SAFEGUARDS.

Platform at gas-relief valve on hot-blast main. Ventilator over valve if inside cast house.

Check valve in mixer pipe.

Butterfly valve in cold-blast main on furnace side of snort valve.

Oil drain from cold-blast line at stoves.

Two spring-loaded blast-relief valves on cold-blast line, adjusted to blow off at any desired pressure.

Steam connection to cold-blast line at engine room.

Oil drain from cold-blast line at engine room.

Means of opening snort valve from outside of cast house.

Muffler on snort valve.

ENGINE-ROOM SAFEGUARDS.

A whistle of distinct and different sound for each furnace blown from the same room, and a number corresponding to the furnace displayed simultaneously with the sounding of the whistle.

Device to prevent engines from turning over backward owing to pressure of blast.

Guarding of fly-ball governors, rocker arms, connecting and tail rods, cranks, and air-inlet valves.

Guarding of flywheels with 5-foot wire fencing, carried over the bearings.

Three-rope drive of governor.

Emergency stop.

Relief valve at each end of air tub.

Automatic oiling system.

Checkered platforms, with standard railings and nonslip safety treads on the steps from gallery to gallery of engines.

Air intake from outside or guarded with wire netting.

Blocking for cylinder and flywheels.

Oxygen breathing apparatus at gas-engine rooms.

Forced ventilation of gas-engine basement.

SAFEGUARDS IN CONNECTION WITH SLAG DISPOSAL.

Valve for granulating-pit sprays placed at safe distance from runners and pits.

Tracks for cinder ladles covered at cast house.

Steam or air dumping of cinder ladle at dump. Devices on cinder ladles to prevent overturning in transit or loading.

Means of extinguishing burning clothing at cinder dump.

Cab on cinder crane so placed that it does not come over pit during travel of crane.

Prevention of steam from granulating pit from getting into cast house.

SAFEGUARDS FOR LADDERS.

Keeping portable ladders at storeroom, numbering them, and issuing them upon a return requisition.

Equipping portable ladders with metal spurs at the bottom and hooks at the top.

Keeping rungs of permanent ladder at least 8 inches from walls, columns, or stacks.

Providing cages for all permanent ladders more than 18 feet high.

SAFEGUARDS FOR SCAFFOLDS.

Removal of every piece of loose lumber from about plant.

Selecting scaffold lumber as lumber is shipped into plant, testing it for strength, marking it, and setting it aside for scaffold use only.

Placing scaffold lumber in store yard where it can be taken out and used only by authorized carpenters.

SAFEGUARDS FOR RIGGING MATERIAL.

Chains, cable, rope, and slings given out only by tool man or boss rigger, who has tables of safe working loads.

Inspection of material by boss rigger or assistant every time it is returned.

Destruction of defective rigging material as soon as dangerous condition is ascertained.

Safety belts and lines, with wrought-iron snaps, buckles, and rings.

SAFEGUARDS FOR YARDS AND TRACKS.

Elimination or guarding of all holes and depressions, close clearances, dangerous corners and doorways, walks and paths beneath cranes, and eaves of roofs near ladles being transported, poured, or loaded.

SAFETY APPLIANCES.

General dispensary for first-aid treatment.

Stretchers at various points along trestles and in stock house, cast house, boiler and blowing room, shops, ladle house, and around pig machine.

Resuscitation apparatus.

One or more half-hour oxygen breathing apparatus.

Regular drill in first-aid measures and use of breathing apparatus, stretchers, and bandages.

MISCELLANEOUS SAFEGUARDS.

Handles on crane hooks.

Wearing goggles or masks in handling slag and metal.

Box-toe shoes for men handling material.

Portable railings for manholes and trapdoors.

Portable runways, cleated, and 24 inches wide, in place of miscellaneous plank.

Metal covers on all ditches and sewers, especially hot-water sewers.

Portable danger signs for use when any work presents danger to men in the vicinity, or for temporarily hazardous furnace or other conditions.

The foregoing list comprises only a small part of the guards used at blast furnaces. Those mentioned are considered desirable by one or more organizations. In addition to these, the market is full of guards of all kinds for protecting all sorts of places and for all the



A. SAFETY ENTRANCE TO BLAST-FURNACE PLANT.

The subway extends under all tracks inside yard.



B. WORKMAN IN DANGEROUS POSITION ON CAR. IS LIABLE TO BE STRUCK BY MAGNET.



A. LOCOMOTIVE CRANE, SHOWING GUARDS.

A, Rotating frame; B, swinging angle-iron bumper; C, floor plates; D, shield; E, hand iron; F, step; G, load indicator; H, extension for automatic coupler; J, coupler lever; K, warning sign.



B. GUARDS OVER FRICTION PULLEYS AND GEARS ON ORE POCKETS.

Note safety lines for stopping machinery.

machinery in the plant. Money may be spent without limit, every recommendation may be followed, and every appliance purchased, but if the installation of these appliances is not accompanied by a realization on the part of the force that they must be used, their full efficiency is not realized. Few guards or safety appliances can be made foolproof. To insure not only the maximum mechanical results from the safeguard but also the full benefit of the investment in the way of decreased accidents, the installation of safeguards has to be made the basis of further time and effort in educating foremen and men to think about personal responsibility and personal safety, in emphasizing the close relation of doing work efficiently to doing it safely, in continuing efforts to prevent hazardous work being done carelessly or routine work being done thoughtlessly, and in encouraging the cooperation of the men in correcting incorrect or dangerous practices, improving hazardous conditions, and suggesting plant betterments. The safety committee of workmen is generally accepted as being the best means of accomplishing the desired results.

Below is shown a warning sign used by one large blast furnace. The sign is made of blue and white enamel and gives general instructions to be observed around the blast furnaces, as well as mentioning special precautions to be taken when working on apparatus controlled by steam.

NOTICE -
TO
BLAST-FURNACE EMPLOYEES.

DO NOT GO UP STAIRWAYS TO STOVE PLATFORM OR FURNACE TOPS WITHOUT PERMISSION FROM THE BLOWER.

NEVER GO ONTO FURNACE TOPS ALONE.

DO NOT GO ABOVE THE RECEIVING HOPPER OF ANY FURNACE WITHOUT LOCKING AND TAGGING THE LEVERS OPERATING THE BELLS.

DO NOT GO ON THE SKIP-CAR TRACKS, IN THE SKIP PITS, OR ANY PLACE WHERE THE SKIP CAR CAN HIT YOU WITHOUT LOCKING AND TAGGING THE MAIN CIRCUIT BREAKER ON THE HOISTING ENGINE.

DO NOT WORK ON THE COKE-BIN DOOR CYLINDERS WITHOUT LOCKING AND TAGGING THE LEVERS OPERATING THEM.

DO NOT ENTER ANY STOVE NOR CLEAN STOVES FROM THE UPPER PLATFORM WITHOUT LOCKING AND TAGGING THE COLD-BLAST VALVE, HOT-BLAST VALVE, CHIMNEY VALVE, AND GAS VALVE.

DO NOT GO NEAR THE GAS DOOR OF A STOVE WHEN ON GAS.

DO NOT TIGHTEN NUTS ON A STOVE DOOR WHEN THE STOVE IS ON BLAST.

DO NOT USE TELEPHONE WHEN A MISUNDERSTANDING CAN CAUSE AN ACCIDENT.

BLANK COMPANY.

FEBRUARY 1, 1913.

Plate XII, *A*, shows a safety entrance to the blast furnace provided at one plant. The subway extends under all of the tracks inside the yard.

Plate XII, *B*, shows the need of care on the part of men climbing in and out of cars, in order to avoid being struck by magnets, loads, etc. The man is shown to be in imminent danger of being struck by a crane.

An effectively guarded locomotive crane is shown in Plate XIII, *A*. The rotating frame *A* has been elevated to give a clearance of 8 or 9 inches over the deck of the truck. The swinging angle-iron bumper *B* serves as a warning by striking any one standing or leaning on the truck. The floor plates *C* are extended to the full width of the truck. The gears, clutches, and engines are guarded by the shields *D*. Grab irons *E* and steps *F* are suitably placed for the safety of men getting on or off the crane. An automatic load indicator is shown at *G*. At *H* are shown structural extensions to both ends of the frame, to which automatic car couplers are attached, allowing the crane to be coupled to cars without danger. A coupler-operating lever, *J*, and a warning sign, *K*, are also shown.

Plate XIII, *B*, shows guards over friction pulleys and gears on blast-furnace ore pockets. Safety lines with handles to be pulled to stop machinery are also shown. The rope pulls out a switch and, cutting off all power, sets a brake, stopping the machinery.

A thoroughly guarded platform and stairway leading from the cab of a skull-cracker crane to the crane bridge is shown in Plate XIV, *A*.

Plate XIV, *B*, shows blast-furnace boiler counterweighted doors which open inward and upward, eliminating danger from any fire-box explosion.

Plate XV, *A*, shows a high-tension junction pole, equipped with ladder and with platform at the top and guard plates on the sides of the ladder to protect a man while passing up through the wires.

Plate XV, *B*, shows a heavy wire mesh guard at the oil switch and a lightning arrester in a blast-furnace power house where the electric current is 6,600 volts.

A well-guarded Bessemer mixer crane is shown in Plate XVI, *A*. A safety limit switch, *A*, prevents overtravel of the hoist. The carbon contacts, *B*, are brought together when the hook block raises the hinged arm of the switch. When the contact is made the armature is short-circuited, stopping the motor. A safety grease cup, *C*, and a shield, *D*, over the sheaves are shown.

Plate XVI, *B*, shows boxes used at one large blast-furnace plant to contain woolen blankets for use in case of fire to employees' clothing. The round box is a portable moisture-proof can used where naphtha oil is applied to pipe, rails, or other material with portable



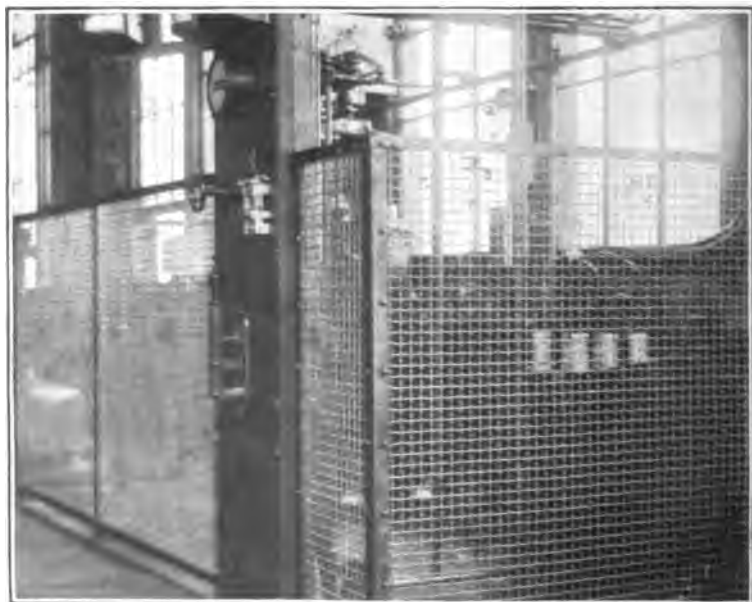
A. GUARDED PLATFORM AND STAIRWAY FROM CRANE CAB TO BRIDGE.



B. COUNTERWEIGHTED DOORS ON BLAST-FURNACE BOILER; DOORS OPEN INWARD AND UPWARD.



A. PLATFORM AND LADDER WITH GUARD PLATES, ON HIGH-TENSION JUNCTION POLE.



B. WIRE-MESH GUARD AT OIL SWITCH, AND LIGHTNING ARRESTER USED AT BLAST-FURNACE POWER HOUSE.

spray pumps. Boxes like the square one shown are permanently located at numerous places throughout the plant where there is danger of the clothing catching on fire. The door of the box automatically locks from the inside, making it necessary to break the glass to remove the blanket. The broken glass falls through a slot in the bottom of the box. The hook shown is for use in breaking the glass.

Plate VIII, *B*, and Plate VIII, *C*, show a safety hook for use where hoisting is done in a confined space, with consequent danger that the hoisting bucket may catch on walls and become disengaged. This hook is used by bricklayers on stacks, blast-furnace stoves, etc., and by boilermakers, machinists, and others. Plate VIII, *B*, shows the hook closed. The sleeve *A* fits into the notch *B* and locks the hook, making it impossible for the bucket to become disengaged from the hook and fall on workmen, either when the bucket is being hoisted with a load or lowered when empty. Plate VIII, *C*, shows the hook open. The latch *C* drops and is locked automatically.

An efficient type of car s'ifter is shown in Plate XI, *C*. A four-cornered piece of hardened steel is inserted in the forging. When one corner of the steel becomes worn the steel is driven out, turned around, and another corner is used.

MEANS OF ACCIDENT PREVENTION.

RELATIVE VALUES OF DIFFERENT MEASURES.

Few attempts have been made to valuate various methods of accident prevention, but one company has expressed the relative values of methods of preventive measures as follows:

Relative values of accident-prevention measures.

Preventive measure.	Value, per cent.
Organization:	
Attitude of officers.....	20
Safety committees.....	20
Inspection by workmen.....	5
	— 45
Education:	
Instruction of men.....	15
Prizes.....	9
Signs.....	3
Lectures.....	3
	— 30
Safeguarding:	
Safety devices.....	17
Cleanliness.....	5
Lighting.....	3
	— 25

These figures are representative of many years' experience, but further experience will undoubtedly lead to further changes. In fact, experience with financial rewards to foremen and men for results in accident reduction indicates that "prizes" should receive a much higher valuation than is assigned. Comparative accident reduction over two years in which a bonus was paid individual employees for accident prevention shows comparative rates of 100, 54.9, and 24.6.

PLANS FOR PAYING BONUSES.

The bonus for a foreman is paid in a variety of ways. One scheme is to fix the bonus at a certain sum per man on the foreman's gang. At \$2 per man of a 20-man gang the maximum payment would be \$40 if no accidents occurred. To fix a "bogey" rate, the accident rate for the preceding one, two, or three years may be taken. Assuming a rate fixed at 20 per cent to start, and that at the end of the period it has been reduced to 12 per cent, the bonus will be twelve-twentieths of \$40, or \$24. The period at the end of which the award is made may vary from six months to a year. Reductions of accidents of 45 to 70 per cent have been effected by bonus payments.

Another scheme of bonus awarding is as follows: A plant is divided into districts and a committee in each is appointed to serve for a period of two months. A datum line showing the frequency of prior accidents in each district, in units per 100 men employed, is established. Each district is expected to equal or excel its previous record, the records not being set against each other, but the record of each being wholly within itself. Each 60-day record is merged into the previous total, and a new record is automatically set. For each man of a committee that equals or excels its own record during a 60-day turn there is provided a \$10 prize. In addition, the committee making the greatest improvement is granted a double prize, or \$20 to each man. This plan of cash reward for accident prevention reduced accidents by approximately 60 per cent in one year.

The obvious objection to these plans is that as accidents decrease there will be diminishing returns to the foremen or committeemen. To offset this situation the basing rate should be increased by 10 to 40 per cent at the commencement of each period.

INSTRUCTION OF MEN AND LECTURES.

If "instruction of men" and "lectures" be grouped as preventive means the rating is 18 points. "Lectures" refers to a policy of having a foreman get his men together and give them a brief talk on the importance of safe practices, giving emphasis to the policy of the management, and mentioning specific instances of injuries and the means by which they are to be avoided. This practice is one way of recognizing that the foreman is the main factor in determining the



A. CRANE HOIST, SHOWING SAFEGUARDS.

R, Safety limit switch; B, carbon contacts brought together when hook block raises arm of switch, short-circuiting motor; C, safety grease cup; D, shield over sheaves.



B. CONTAINERS FOR WOOLEN BLANKETS FOR EXTINGUISHING BURNING CLOTHING.

Can is portable. Box is placed at danger point; has automatic inside lock, glass front, and hook for breaking glass; cross is green enamel.

quality and quantity of work done by the men under him. After a plant has been constructed according to the best design and has been provided with adequate equipment, the output, the upkeep of the machinery, and the efficiency of the men, including their freedom from injuries, depend on the foreman. Perhaps the most difficult problem in accident prevention is to arouse among the foremen a genuine interest in efforts to avoid accidents. Often certain risks are realized but are regarded as necessarily incident to the work. This does not mean that foremen are indifferent to the safety of their men, but that their first concern is production. If they realize that accident prevention decreases the cost of production and does not conflict with the work, but that it should on the contrary increase the efficiency of the men as many accidents are due to inefficient methods of work, they do not lack interest. Given this interest, instruction of their men by talks, admonitions when the men are seen taking uncalled-for risks or hazards, and adequate supervision follow automatically.

Other "lectures" are motion pictures pertinent to plant safety, stereoptican lectures, and conferences. At some plants the management hires a large hall for a meeting place, provides cigars and light refreshments, invites all the men the hall will contain, and asks a free discussion of methods, advantages, and results of safety work. When a bonus scheme is in effect a public meeting is made an occasion on which to present prizes.

SIGNS, CLEANLINESS, AND LIGHTING.

Signs, cleanliness, and lighting are given a rating of 3, 5, and 3 per cent, or a total of 13 per cent. These features, if defective, can be corrected almost immediately. The use of signs calls for considerable discretion. In plants giving much attention to safeguarding, it has been found that when signs are used too lavishly employees become so familiar with such warnings that the desired purpose is defeated. About blast furnaces permanent signs may be used at the following places without fear of overplacarding the plant:

1. Steps to top of furnace and stoves.
2. Steps to gas-engine basements and to tops of boilers.
3. Beneath pig-machine strands or chains.
4. At railroad crossings and track approaches.
5. At electric conductors carrying more than 250 volts.
6. At close clearances along tracks.
7. Under dust catchers.
8. About skull cracker.

It is the consensus of opinion that the English word "danger" in white on a red sign carries to the foreigner who understands no English exactly the same significance as to English-speaking people,

because he can readily understand that it is used only to indicate places of peril. Such permanent signs should be lighted at night. Small signs that can be attached to electric switches, and to water, steam, vacuum, blow-off, or gas valves should also be provided. These should have a white "danger" on a red background. Portable signs to be placed under men working above, or where men are chipping, cutting rivets, or breaking scrap in the cast house or the ladle house, or shooting salamanders or frozen ore, should be provided.

Cleanliness is characteristic of most blast-furnace plants, and emphasis on this point would be largely misplaced. Furnace work being a rough trade and productive of much rubbish, brickbats, slag, and scrap lumber and iron, it is necessary to keep everlastingly at the work of cleaning up, and as a rule, yards, cast houses, pavements, and houses are noteworthy neat. The few exceptions only go to prove the rule.

ACCIDENTS AND ACCIDENT PREVENTION AS RELATED TO THE EMPLOYEE.

LOSS OF TIME.

Anyone who has worked with or has had the direction of workmen realizes that one of their chief fears is that they will lose time and wages as the result of time keeping or other clerical errors, plant shutdown, or enforced absence. With average full-time weekly earnings of \$12.43 for laborers to \$19.13 for keepers in the furnace industry, an average loss for injury of \$17.95 over noncompensated periods, or \$50.24 over average compensated periods, represents an impairment of income that is sometimes accompanied with hardships and deprivations. It is common to see painfully injured men bearing their injuries stoically and bravely but seemingly undergoing much mental suffering on account of the uncertain loss of time and income facing them.

Malingering is an act of which men are sometimes accused. On the other hand, many injuries terminate long after the normal period, owing to lack of proper medical care caused by unsystematic follow-up arrangements, neglect on the part of the man himself, or a lack of system in returning the injured man to work when the condition of the injury is such that the probability of additional loss is exceedingly remote. The average duration of all injuries is 17 days. A large steel works having an excellent follow-up system and medical treatment reports 12 days as an average of all injuries when more than a day's lost time is necessary. Infections ordinarily comprise 1 in 39 injuries, but with adequate medical treatment they are reduced to 1 in 900.

The 14 days' exemption period provided by law is designed to prevent malingering and to put upon the employee a sufficient part of the burden of accident cost so that he has a strong incentive for returning to work promptly. This provision of law is essentially a safeguard provided for the employer against unjust claims. It is based on the probable malingering of a minority of employees, and it works a real hardship on the majority. Under these circumstances one of the best means of demonstrating the usefulness of accident-prevention means is for plant managements to undertake measures that will return men to work at the earliest possible hour.

When employees realize that every practicable attention is being given to this phase they will tend to accept without reservation the intention of the management to undertake safety work for the good of the employee as well as that of the employer. There is not infrequently resentment against lost time imposed by the physicians' order. Until any possible grounds for such resentment are removed, it is almost useless to talk safety to men on the basis of loss of time. With satisfactory conditions, loss of time is a strong point in impressing on employees the importance of personal care.

DISCIPLINE.

Imposition of discipline or fines for infraction of a safety rule or for carelessness is a cause of resentment among employees. Intelligent carefulness all the time is beyond the power of most men, of any station, especially among a large proportion of furnace employees. Discipline for trivial offenses, for mishaps as a result of desultory and untrained methods of work, or as aggravated by personal antipathy of a foreman for a workman injures the cause of safety among the men more than it can possibly help. So far as is known, fines have been imposed at only one plant. Layoffs or discharges are in the hands of the foremen and the superintendent at most plants, the suggestion or recommendation of the safety inspector being mandatory at only a few plants. Two plants settle the question of discipline by a "court of inquiry," where all concerned in an accident and all witnesses are invited to give their version or make defense. Thus the man is judged to his own satisfaction by his associates. Wherever foremen of gangs will surrender their prerogative of discipline for infractions of safety rules, the court mentioned removes another obstacle in safety work.

USE OF INTOXICANTS.

In few of the accidents reviewed was drunkenness given as the primary or contributing cause of injury. This is probably due to two facts: First, drunken men are without exception turned back from

all plant entrances; and, second, men coming to work after alcoholic excess are, though not intoxicated, either sleepy and dull, ill, or reckless. Although a man in such condition has neither mental or physical balance, and is more susceptible to accident; in event of accident the part played by alcoholism is determined only as result of time and effort in tracing back a suspicion to a certainty, an effort comparatively seldom made.

In the eradication of personal habits or excesses detrimental to the safety of the men, alcoholism must at times be given attention, but the matter is to be approached with caution, welfare effort in this direction being often resented as an encroachment on personal liberty. The fairness of such work can be impressed upon men by making clear that the compensation act means the discharge of a man who gets hurt too frequently, because he is a bad risk to the company, that such a man eventually will not be able to keep himself in a job, and that alcohol in excess will surely bring any man to such a state. Fortunately, alcoholism has not been an extremely serious factor in accident-prevention work at most furnace plants, because an habitual, excessive drinker soon loses his job about a furnace through too frequent absence or too frequent intoxication.

ATTITUDE OF FOREMEN.

The attitude of foremen, subforemen, and "straw bosses" toward safety work determines largely how the men view the work. Some foremen think that a man who is clearly careful must be too careful, that he does not do as much work, or does not do the set work as quickly as other less careful men. As the foreman is in contact with the men, and sets the policy for the manner in which work is done, the success of efforts to interest the men rests largely with him. Nothing can so effectively smother safety-committee work, men's suggestions, bulletin-board publicity, or stimulation by prizes, as a foreman's conflict with suggested safeguards or changes in methods of work. The management of plants should make it evident that the safety of the men in each foreman's gang is of at least equal importance to the efficiency or productiveness of the gang by the way in which the foreman is paid or promoted. Foremen can use the same incentives to induce carefulness that they use to induce efficiency. It follows that workmen will promptly assume their share of the work and responsibility.

PHYSICAL EXAMINATION.

The physical examination of workmen as a means to promote industrial safety is much used in large plants. It has met with opposition from workmen who fear the information derived from the

examination will be used to their disadvantage. Physical examination should be made with a view to fitting a man's work to his mental and physical capacity, not to weed out defectives from the employees unless the defect results from a clear-cut case of bad personal habits.

Physical examination of workmen has probably resulted in more good than in hardship to workmen, particularly those who are uneducated in American schools. The tendency of these men is to neglect physical weakness until they become permanently impaired. If as a result of physical examination men afflicted with unsound hearts, hernia, or deafness are given work less hazardous to themselves and to others, or if the examination discloses incipient tuberculosis, defective sight, varicose veins, or infectious social diseases, with the result that the defects are cured, the examination should not meet with objections from any except the small percentage who object on the ground of paternalism or have religious or personal reasons for declining examination. Physical examinations conducted in the right spirit and at proper intervals will bring greater health and safety to workmen.

RELATIVE HARDSHIP.

It goes without saying that safety work is of advantage to both employer and employee. Yet, in seriousness of burden, caused by unsafe conditions, the employer escapes lightly. With an accident rate of 500 per 1,000 men, the estimated cost would be only \$0.028 per ton of pig iron. The employee, on the other hand, stands one chance in two of partial impairment of earning power, amounting, on the average, to 2 to 3 per cent of his yearly income, not to mention his liability to loss in earning power of thousands of dollars by permanent disability. So it is true that accident-prevention work on the part of the employer must rest on more than a desire for a good showing on the profit side of the ledger.

Another important inducement for the employer to provide the maximum of safety is embraced in the general labor situation. The mass of labor about blast-furnace plants, except in one district, has been composed of immigrants, and the standard or level of wages has been the lowest of any department of the steel industry.

It is a matter of record that Americans born of native or foreign parents are not attracted in large numbers to the industry. It is a live question as to whether it will be possible to continue to attract the immigrants, or whether with future changes of conditions in their native lands men will not cease to emigrate. It is more than likely that a future problem of the furnace industry will be to attract immigrants, or to attract American-born laborers, by making employment more inviting. The problem will be partly technical,

involving the character of work to be required, and partly sociological, involving the conditions under which the work is performed. Development of safety measures, such as improvement of working conditions, that is, providing adequate sanitation, locker, bathing, and lunch rooms, and drinking-water facilities; encouraging better home life, conserving health; eliminating so far as possible fatigue and occupational diseases, and lessening the causes of dusts, gases, vapors, and fumes, excessive heat and cold, glaring lights will go far to supplement the work set for technical men in lightening and making more attractive the work performed about furnace plants. Thousands of dollars have been spent in this work, and the tendency to make improvements is constantly growing. The record of many plants in this respect is one of progress and recognition of their responsibilities, and there has been almost more betterment in conditions of employment in the past 5 years than in the preceding 50.

Safety work, therefore, should rest on a very secure conviction in the minds of both employers and employees that it is a decided necessity.

SAFETY-COMMITTEE WORK.

IMPORTANCE OF SAFETY COMMITTEE.

Throughout this report repeated mention has been made of the necessity of persistent and continued personal efforts on the part of foremen to correct careless and unskillful methods of work. The writer's observations lead him to believe that herein lies the best hope of reducing the large number of accidents that no safeguarding can eliminate. To anyone who has studied the types of accidents cited a similar thought must have occurred. In concluding the report with a discussion of some aspects of safety-committee work, emphasis is again laid on the importance of daily personal work by foremen to supplement the efforts of safety committees.

The methods of organizing safety-committee work are known to all officials of furnace establishments and need not be gone into here. At the time this report was prepared all blast-furnace plants operated in conjunction with steel plants were utilizing safety committees of men, or of foremen and men, as an aid in reducing the number of accidents, and the work of the committees was meeting with success. However, few isolated plants had found the safety committee to be an unqualified success.

SAFETY COMMITTEE AT A SMALL PLANT.

A small plant is under a handicap in undertaking safety-committee work. For one reason there are fewer accidents in a force of 100 men than in one of 400, so that the safety committee at the larger

plant has a bigger task and one that engages its respect and interest at the start. There is always some practice, condition, or equipment that has caused a more or less severe accident. This they can talk about and correct with some satisfaction to themselves. At the expiration of their service their endeavors are represented by a respectable amount of work accomplished. The committee may have an average of five to seven accidents to engage its attention during one month. These accidents give the committee something definite to do; there is something concrete to correct; unskillful, thoughtless, or incorrect practices and hazardous conditions can be pointed out. The committee at the small plant, with exactly the same rate of accidents as at the large plant, will have to consider an average of only one or two accidents per month. Even with the utmost stress, the safety committee at the small plant can not make the significance of one or two accidents as striking to the individual man they are trying to reach as if five to seven had happened.

Experience shows that the average committee at a small plant expends its efforts mainly in providing mechanical safeguards, which is the logical line of endeavor at the inception of a safety campaign. However, the tendency is to persist in this field, which gradually narrows, so that the committee, finding its efforts to accomplish useful safeguarding becoming circumscribed, sooner or later begins to consider its work futile and loses interest long before the actual hazards of the plant, as indicated by the accident rate, have been reduced to the lowest possible figure and kept there. The work loses momentum while improvements are still needed in methods of work, scaffold practice, condition of tools, upkeep of rigging, and other details, and while the workmen only partly realize the full significance of accident-prevention efforts.

To make safety-committee work at a small plant permanent and of everyday usefulness, experience at a few small plants where it has been a success indicates that the committee has to be given a larger field than that assigned to committees at large plants where the work supplements the safety work of the whole steel plant.

One successful plan is to make the safety committee an efficiency committee, putting up to the committee of foremen the question of plant economies. Every two weeks the committee has a meeting, at which minutes are kept. The superintendent meets with the committee as often as he has any improvements, readjustment of force, or similar matter in mind and talks it over with the foremen. When an unusual job is to be undertaken, it is discussed from beginning to end, and all possible contingencies are brought out by the men from the various gangs or departments, and after the best way to handle the work is apparent the sense of the meeting is expressed

in a "work order," which gives explicit directions as to how the work is to be done, who is to direct it, and the precautions to be taken. Economies in storeroom supplies, the handling of material, the labor problems, and similar matters are discussed. The foremen constituting such a committee come to realize the value of cooperative effort, and in connection with the committee work the men advance valuable ideas. The plan is not such a radical departure from accepted plant organization, as every superintendent discusses personally with his foremen all unusual developments in the foreman's work. Referring such matters to the plant committee simply applies to a small plant the idea of a general conference, largely used in big organizations.

OUTLINE OF FOLLOW-UP WORK.

To insure a permanent and active plant committee the superintendent may outline for the committee a broad field of accident-prevention activities. The provision of safeguards can be largely left to the initiative of the committee, with occasional suggestions as the need for them comes up from time to time. However, guidance may be needed in outlining a follow-up system of correcting unsafe conditions and practices. The typical outline given below may be useful. For putting such follow-up systems into practice a form should be prepared, to be signed at each plant inspection by the committee men. They should be given authority to correct or to ask foremen to correct any dangerous condition or practice that is not in conformity with logical requirements for safety.

Outline of follow-up work by safety committee.

[List of items to be observed in promoting general safety about blast-furnace plant.]

TRESTLE.

1. Condition of walks, railings, and crossovers.
2. Condition of tools and manner of using tools.
3. Loose material or tools along walks.
4. Use of safety belts in cars and bins.
5. Carefulness of larry-car, ore-bridge, and train operators.

Stock House.

1. Condition of fenders, lights, and gongs on scale cars.
2. Condition of bin chutes, freedom from lumps liable to fall, and practice in drawing material.
3. Condition of barrows, tools, and lights.
4. Use of goggles and carefulness in breaking lumps, drawing dusty coke, and running scale car.

Hoist.

1. Replacement of guards on hoist, engine, and motor.
2. Use of locks or tags on operating levers and switches.

3. Safety of work in cleaning skip pit and in oiling hoist, skip, and engine or motor.

4. Observation of rules for oiling and inspecting on top.

CAST HOUSE.

1. Condition of railing, steps, and floors.
2. Filling of ladles (not more than within 6 inches of the top).
3. Putting water on trough after cast.
4. Use of shields at tapping hole, cinder notch, and shutters.
5. Use of goggles and masks at cast or flush.
6. Filling of cinder ladles (dryness of ladles and not more than within 6 inches of the top).
7. Condition of tools (no mushroomed heads, cracked handles, or loose sledges).
8. Cleanliness of cast-house floor, and replacement of tools in proper position.
9. Men working alone above bustle pipe.
10. Tight bridle springs.
11. Condition of shoes and clothing of cast-house crew.

STOVES.

1. Locking or tagging of gas burners and hot-blast valve when men are inside cleaning stove.
2. Leaky gas burners.
3. Men standing about in front of stove doors.
4. Handling of gas in lighting and in cleaning wells.

BOILERS.

1. Locking or tagging of burners and blow-off valves when boilers are being cleaned.
2. Water-gage glass protection and manner of using glass.
3. Condition of steam hose.
4. Covering of blow-off and other ditches.
5. Men working alone above boiler settings.
6. Leaky burners.
7. Lighting.

GAS MAINS.

1. Proper handling and wetting down of flue dust.
2. Nonuse of clamps on bells and downlegs.
3. Condition of water and steam lines to gas mains and to water-seal valves.

ENGINE ROOM.

1. Accompany oilers and wipers to observe whether there are hazardous conditions and practices.
2. Inspect platforms and steps for excess oil and slippery conditions.
3. Inspect railings.
4. Men working in gas-engine cellar.
5. Condition of oxygen apparatus.

PIG MACHINE AND LADLE HOUSE.

1. Condition of shoes and clothing of men.
2. Length of poles used.
3. Use of goggles.
4. Method of work at quenching pit.

5. Condition of tools.
6. Absence of rubbish under foot.

TRANSPORTATION.

1. Unauthorized persons on railroad equipment.
2. Use of danger sign by men working in, under, or about cars.
3. Kicking of cars across drives, roads, passageways, or into dead-end tracks.
4. Proper use of bell and whistle.

YARDS AND GENERAL LABOR.

1. Piling of brick, lumber, castings, and pig iron.
2. Use of ordinary plank for runways.
3. Safety of work at cinder dump.
4. Yard cleanliness.
5. Guarding of holes, ditches, etc.

CRANES, HOISTS, AND RIGGING.

1. Method of work in booking on.
2. Methods of work in cars with buckets and magnets.
3. Leaving balls of buckets standing up.
4. Use of proper slings for hoisting material.
5. Inspection of block and tackle—frequency of inspection, care while being stored, disposition of defective material.
6. Condition of tool-room tools and supplies.

LADDERS AND SCAFFOLDS.

1. Selection and suitability of scaffold material.
2. Condition of existing scaffolds.
3. Scaffolds constructed by men other than carpenters.
4. Eliminate miscellaneous makeshift ladders and scaffolds.
5. Condition of permanent and portable ladders.
6. Methods of using ladders.

GENERAL.

1. Condition of previously installed safeguards.
2. Condition of electric-light and motor-extension cord and cable.
3. Condition and legibility of danger signs.
4. Use of portable danger signs.
5. Condition of stretchers.
6. Condition of and training in resuscitation and breathing apparatus.

[To the above may be added the outstanding causes of accident as shown on pages 10–11.]

NEED OF REPORTING MINOR INJURIES.

The men comprising the safety committee should make special effort to get every man suffering injury to report to the first-aid room for treatment. Any injury, however slight, should be reported, including every bruise, cut, scratch, abrasion, strain, foreign body in the eye, or blister. If all such injuries are reported, in the average small plant 10 to 20 injuries will be treated each month. In some large plants where practically every injury is reported, 10 to 15 per

cent of the force report at the hospital every month for treatment. Most of the injuries are trivial, but many of them might become serious through infection if neglected. The safety committee of a small plant can do much effective work in helping its own plant to reach the same record in the reporting of trivial injuries. In so doing, not only will the number of infections become markedly lessened and men be saved considerable lost time, but the safety committee will at the same time place itself in an increasingly stronger position to emphasize the importance of its work by the number of accidents to which it can refer.

GLOSSARY.

- AIR TUB.** The cylinder on blowing engine which pumps the blast, "wind," or "air."
- BARRING SCRAP.** Prying adhering scrap from runners, ladles, or skimmers.
- BIN FEEDER.** Man who rods or bars ore that sticks in going through bin doors.
- BIN MAN.** One who pokes down ore in bins to keep it moving to the chutes.
- BLEEDER.** An escape valve for gas, at the top of the furnace or along the gas line, to relieve excess pressure.
- BLOWER.** Foreman in charge of operation of furnace and stoves, casting, flushing, stops, repairs, etc. At small plants he is in charge of trestle, stock house, and pig machine as well.
- BLOWING ON THE MONKEY.** Flame blowing from the cinder notch.
- BOIL.** Occurs when molten iron runs over a wet or damp spot or object in runner. The sudden generation of steam often causes an explosion, whereby molten iron is scattered about.
- BOILER SCALER.** Man who cleans scales from boiler tubing.
- BOT.** A cast-iron or forged-steel plug mounted on long steel rod that fits inside of the monkey.
- BOTTING.** Thrusting bot into monkey to stop run of slag during a flush, or when the furnace begins to blow on the monkey.
- BOTTOM FILLER.** Man who fills barrow with ore, coke, or stone, weighs it and places it on cage, or elevator, to be hoisted to top of furnace.
- BURNER, BURNER MAN.** A man who takes care of kilns for roasting ore; largely confined to plants roasting sulphur from Cornwall, Pa., ores.
- CAGER.** Supervises weighing and sequence of sending up components of furnace charge, keeps track of number of rounds, and signals to top filler when it is time to hoist.
- CAR DUMPER.** Mechanical device for tilting a railroad car, hopper, or gondola over sideways to empty it.
- CAR RIDER.** Brakeman or laborer employed to ride on car being hauled up to dumper, or on car pushed from cradle, to apply brake and prevent hard bumping.
- CAST HOUSE.** The roofed (and sometimes inclosed) space in front of and about a blast furnace in which molten iron is run or cast.
- CHECKER ARCHES.** Fire-brick supports built up of arch brick or keys to support the checkerwork on the second, third, or fourth pass of hot-blast stoves.
- CINDER BANK.** Same as cinder dump; indicates an old dump as distinguished from one in use.
- CINDER BREAKOUT.** The slag within the furnace penetrates the brickwork and escapes.
- CINDER DOCK.** A bed containing molds into which, in former practice, slag was run, chilled, and then thrown into cars with forks.
- CINDER DUMP.** Place where slag ladles are emptied.
- CINDER FALL.** See Cinder runner.
- CINDER NOTCH.** The hole, about 5 or 6 feet above iron notch, and 3 feet below tuyères, through which slag is flushed two to three times between casts.

- CINDER PIT.** Large pit filled with water into which molten slag is run and granulated.
- CINDER RUNNER.** Trough carrying slag from skimmer, or cinder notch, to pit or ladle. Also called cinder fall.
- CINDER SNAPPER.** Man who removes cinder skulls from cinder runners between cinder notch or skimmer and pit ladles.
- CLAY GUN.** *See* Mud gun.
- CLEAN-UP MAN.** Usually a pensioner who keeps yard cleaned up, pulls weeds, and does odd jobs.
- CONDUCTOR.** The third rail or electric wire on car dumpers, ore bridge, transfer or larry car. Also conductor on hot metal or roustabout engine.
- COOLER ARCH.** An opening in the tuyère breast of the furnace. The tuyère cooler is placed in it.
- CRAE.** Temporary hoisting winch or permanent winch used to pull ladles, cars, or iron plate in boiler shop, also called "mule" on car dumper.
- CRADLE.** Part of car dumper in which the car rests when it is dumped.
- DEILLING UP.** Preliminary digging out the clay in the tapping hole. This is done with hand, air, or electric drill.
- DUST BELL.** Seal at bottom of dust catcher, dust leg, or water-seal valve; is opened periodically to drain flue dust from the system.
- FURNACE HOLDING-THE-IRON.** Furnace gives much less than normal quantity of iron at casting, although the feed may have been regular. The tapping hole runs iron slowly, and the amount of slag is somewhat scanty. Compare "furnace losing-the-iron."
- HANDYMAN.** At small plants a "Jack of all trades," rigger, millwright, and machinist combined.
- HOIST MAN.** One in charge of running skip and dumping bells.
- HOOK BLOCK.** The lower sheave or block on a crane hoist to which a swivel hook is attached.
- HOOKEE-ON.** Same as Hook-on.
- HOOK-ON.** Man who adjusts cables or chains about objects to be lifted or places hook of crane block in bucket balls, hooks of winches to objects to be moved, etc.
- HOT-BLAST MAN.** *See* Stove tender.
- HOT SPOT.** Any small portion of the furnace shell that is thin and seems likely to burn through.
- IRON FILER.** A laborer who removes iron from cars, sometimes breaks it, and piles and classifies it according to grade.
- KEEPER.** One in charge of opening and closing of tapping hole. Runs iron at cast.
- KISH.** Graphite that separates from pig iron in runners, ladles, molds, or mixers.
- LADLE CHASER.** A man who distributes hot metal in ladles to different operations, keeps hot-metal crew busy to prevent skulling of ladles and delay at mill.
- LADLE-HOUSE MAN.** *See* Ladle liner.
- LADLE LINER.** A man who lines, with brick, loam, and clay, ladle thimbles of hot-metal cars.
- LADLE SKULLER.** Laborer who removes rim and bottom skulls from hot-metal ladle cars.
- LIME MAN.** Attends to slaking lime, running lime water to vats beneath pig-machine mold, and operates lime sprays when machine is running.

- LINESMAN.** In charge of maintenance of light and power electric circuits, sometimes including switchboard; usually inspector takes charge at switchboard.
- LOAM BOX.** A container in which loam is boiled in water by means of a steam pipe. The mixture is used in blast-furnace runners.
- MONKEY.** Small water-cooled bronze casting in cinder-notch cooler through which slag runs from cinder notch when bot is withdrawn.
- MONKEY BOSS.** Man in charge of flushing furnace and of claying up monkey and coolers. Helps on tapping hole also and at cast.
- MUD GUN.** A steam cylinder operating a plunger inside a steel tube 6 inches in diameter. Clay is fed into hopper tube as plunger is worked back and forth and is thus forced into tapping hole, at end of cast.
- MULLIGAN.** A heavy double-hand sledge for breaking runner scrap.
- NIGGERHEAD.** A slip pulley on hoist winches. Rigger takes a few turns of the hoisting rope about the pulley and, by varying tension on the rope, can vary speed of hoist on lowering object with engine running or hoist running.
- ORE BRIDGE.** A large electric gantry type of crane, which, by means of clam-shell bucket, stocks ore or carries it from stock pile to bins or larry car on trestle.
- ORE-BRIDGE BUCKET.** Clamshell grab bucket of 5 to 7½ tons capacity.
- OREGON SLEDGE.** A broad-faced sledge hammer.
- PICKING ROD. PRICKING ROD.** (1) A 1½-inch steel rod, about 20 feet long, used to ram into the tapping hole, while furnace is casting, to dislodge obstructions preventing a good run; also called tapping bar. (2) A bar or rod used for cleaning tuyères and blowpipes.
- PIG STICKER.** Man who punches or knocks pig iron out of chills or molds at pig-casting machine.
- QUENCHING PIT.** A pit filled with water in which graphite, residue of iron, and slag from hot-metal ladles is granulated.
- RAVELING.** Pulling material out of ladle, furnace, or iron trough at tapping hole. Used in different sense at steel works.
- RID-UP RUNNERS.** Ridding up; cleaning up after a cast, when the scrap, slag, and iron are removed from runners, troughs, and skimmers, and they are freshly clayed, loamed, or sanded.
- RIGGER.** Semiskilled employee, duties concerned largely with construction and repair rather than maintenance. Skilled in use of hoist tackle, winches, etc., and usually able to do riveting and to assemble material.
- SAILOR.** A term sometimes employed for rigger, painter, or structural worker.
- SCRAP PICKER.** A man employed on slag dump to pick out lumps and sheets of iron carried to dump in slag ladles.
- SCRAPMAN.** See Scrapper. May also refer to man who breaks and removes heavy scrap in cast house.
- SCRAPPER.** One who removes scrap from bin, cast house, or chute to skip pit and charges the material removed into skip at regular intervals.
- SINTERING MAN.** One in charge of plant for sintering flue dust, or simply an employee there.
- SKIMMER.** Device on tapping-hole trough next furnace by which slag is automatically removed or skimmed from top of iron at cast and diverted to ladles or pit.
- SKIP PIT.** Hole into which skip descends when at bottom of skip incline in stock house, to bring its top below discharge chute of scale car.
- SKULL.** Solidified iron, graphite, and cinder in ladle. Solidified mass in front of tapping hole.
- SKULL CRACKER.** See Skull drop.

- SKULL DROP, SKULL CRACKER.** Device for breaking heavy ladle skulls, iron from messes, or scrap.
- SNORT VALVE.** A butterfly valve opening from cold-blast main to atmosphere. Allows casting at the furnace without shutting down blowing engines. Operated by large wheel or lever in cast house.
- SPELL.** A rest period for crews at furnace, stock house, etc., or a period of work in drilling tapping hole.
- SPLASHER.** A plate lined with fire brick, which is placed over iron trough next to tapping hole to keep down flame that blows from tapping hole during a cast.
- STOCK.** Term applied to the mixture of ore, coke, and limestone charged into the furnace or stored in bins at stock house.
- STOCK DUMPER.** *See* Trestle man.
- STOCK-HOUSE MAN.** General term for anyone working in stock house.
- STOCK UNLOADER.** Laborer who unloads ore, coke, or stone from cars on trestle.
- STOVE TENDER.** One who puts stoves on gas or on blast, regulates temperatures of blast; handles gas at shutdowns; usually watches water from tuyères, plates, etc.
- SWEEPER.** One who cleans the brick pavement between stock house, stoves, and furnace.
- TAPPING BAR.** *See* Picking rod.
- TRANSFER-CAR MAN.** One who operates electric car which transfers ore from ore bridge to ore bin.
- TRESTLE LABORER.** *See* Trestle man.
- TRESTLE MAN.** One who unloads coke, limestone, and ore. Keeps bins poked down.
- TROUGH MAN.** One who takes care of runner at pig-casting machine while iron is being poured from ladle cars; bars out scrap and keeps nose clean; prepares runner for next cast.
- TURN MAN.** An employee who works regularly at various stated occupations six days a week, enabling each of the six men he relieves to get off one day in seven. Cast-house turn man may work as hot-blast man, keeper, first helper, second helper, monkey boss, and cinder snapper.
- TUYÈRE MAN.** Fits up tuyères, plates, and coolers and tests them, to have them ready for replacement in furnace on short notice. Changes bronze when it cracks or is burnt.
- WATER TENDER.** A boiler-house employee attending to feed water of boilers, and usually also to blow-off valves. Frequently is foreman of the house and has charge of gas.
- WELSHMAN.** A heavy steel ring about 3 or 4 inches inside diameter, used in withdrawing a bar which is stuck or frozen in a skull of iron. The ring is placed on the bar, a wedge inserted, and the bar backed out by sledging on the wedge.
- WINDING BAR.** The appliance on drop-bottom cars by which the doors are closed and held tight.

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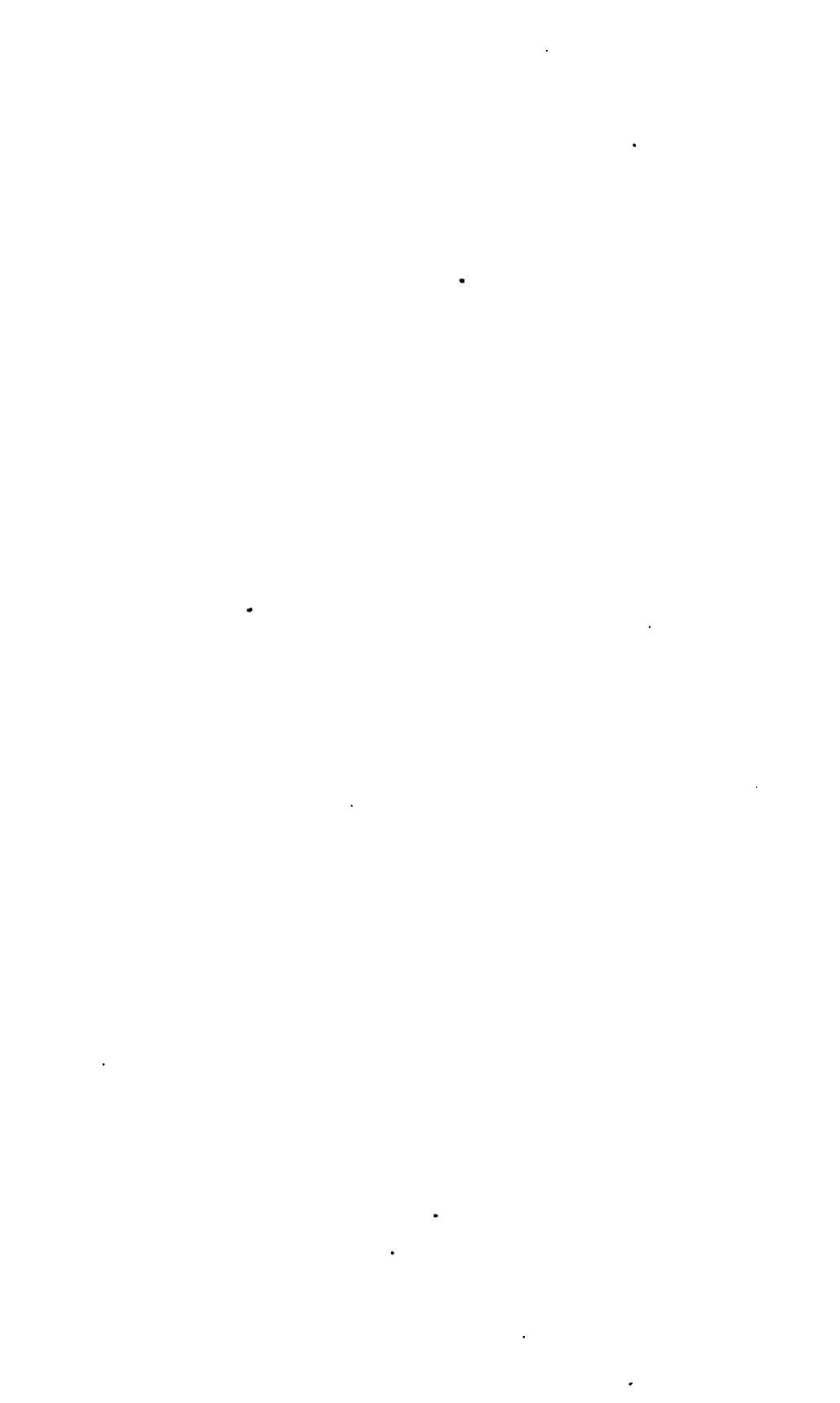
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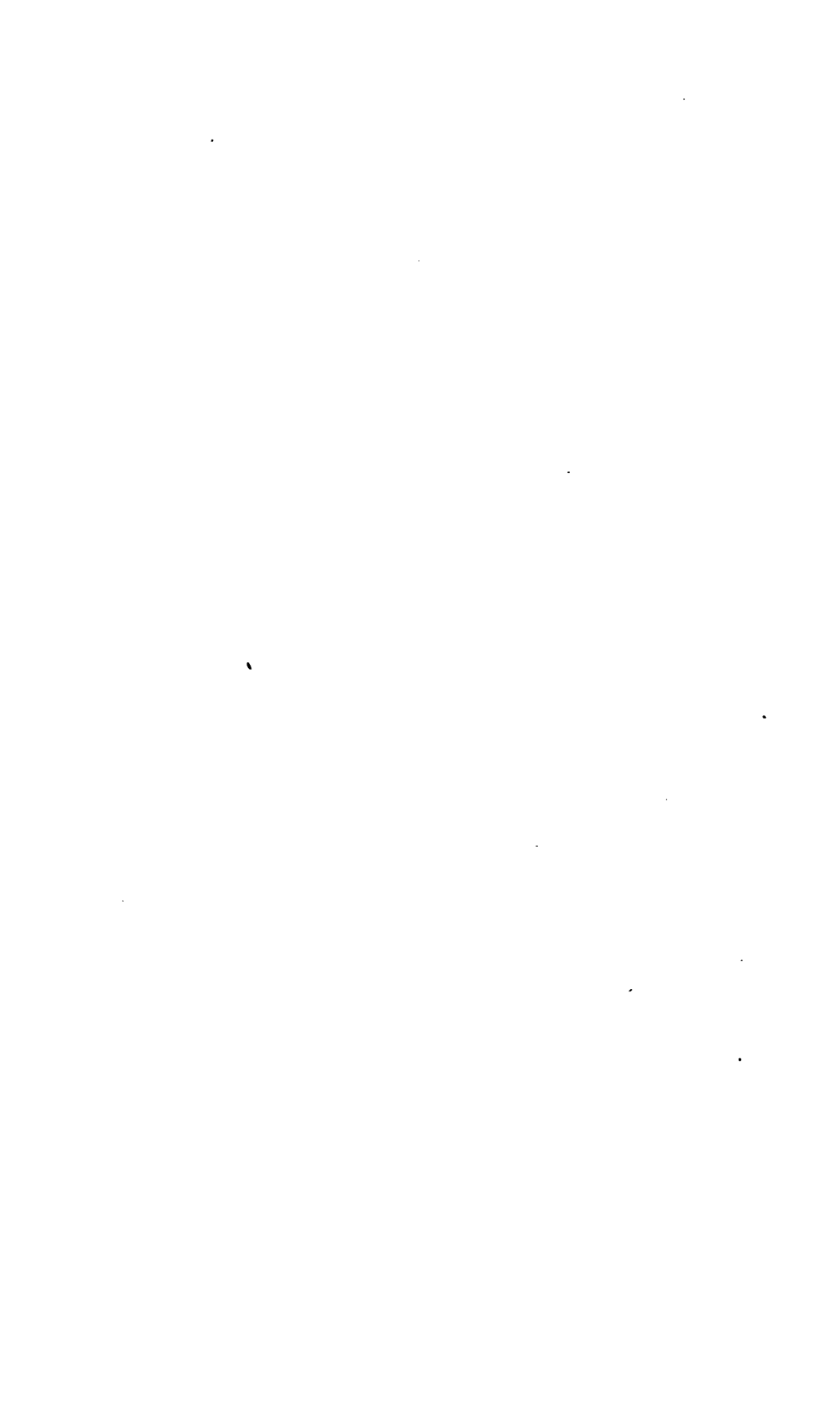
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YEARBOOK OF THE BUREAU OF MINES—1916.

By VAN. H. MANNING.

INTRODUCTION.

Probably no year in the history of the United States showed greater progress in the mineral industries than 1916. Although this progress was undoubtedly stimulated by the war in Europe, which caused extremely high prices for some of the metals, yet the fact remains that mining is being regarded more as a business and less as a speculation, and the benefits of business methods are becoming evident. Practices that were almost universal a few years ago have been largely abolished, the mining public has been brought to recognize the folly of wasteful and dangerous methods, and there is a gratifying unanimity of opinion in regard to the conservation of life and health, as well as of our mineral resources. The people of the United States realize more fully than ever before the extent and value of the country's mineral resources and the possibilities that await intelligent and well-directed efforts at utilization.

In the past, economic reasons caused deposits of valuable metals and industrial materials to lie idle; commercial reasons, many of them not truly economic, led to the waste of valuable constituents in many ores; the recovery of minerals from the ground was too often incomplete and accompanied by uncalculated loss; and the treatment of ores in mills and smelters saved altogether too low a proportion of the metals sought.

Now better methods are being tried on every hand, and the unparalleled record of the year is largely due to the success of these efforts. According to the preliminary estimates of the United States Geological Survey the output of metals and minerals in this country in the calendar year 1916 had a total gross value of slightly more than \$3,000,000,000. For many metals and minerals the figures broke all previous records, the new records of some products, as compared with 1915 figures, being as follows:

Production of certain metals in 1916 as compared with 1915.

Metal.	1915	1916	Increase.
			<i>Per cent.</i>
Manganese, tons.....	9,709	27,000	275
Iron, tons.....	55,403,100	75,500,000	34
Coal, tons.....	531,619,487	597,600,000	12
Cement, barrels.....	86,891,681	94,508,000	6.1
Copper, pounds.....	1,888,000,000	1,928,000,000	1.4
Lead, tons.....	561,639	622,000	10
Zinc, tons.....	606,915	708,000	6.6

The Bureau of Mines, under the terms of its organic act, as amended, is endeavoring to increase the safety and health of workers in the mineral industries and to promote greater efficiency and the prevention of waste in the mining, preparation, and utilization of the mineral resources in the United States. Although difficulties have had to be overcome, and the available funds have been insufficient to meet all the demands made of the bureau, it is believed that the work accomplished by the bureau has kept pace with the progress of the mineral industry as a whole. The increase of efficiency and the lessening of waste in mining are of high importance, and investigations with these ends in view have been furthered to the utmost, but such work must necessarily come second when there is opportunity for work that will increase the safety and health of the workers in mines and mills.

This bulletin describes in some detail the more important work done by the Bureau of Mines during 1916 in efforts to increase safety and efficiency in the mineral industries. The purpose and organization of the bureau and a review of its work for each fiscal year are presented in the annual reports of the director. Those reports are necessarily summarized; they can not give full details of noteworthy experiments nor describe at length new and improved equipment, apparatus, and devices that are being used by the bureau or have been devised by its engineers and chemists. This bulletin gives descriptions of some noteworthy safety devices and discusses in fuller detail than the annual reports the relation of the bureau's work to the general problems of safety and efficiency in the mineral industries and the significance of the results that the bureau has been able to achieve.

In order that the scope of the bureau's activities may be brought out as clearly as possible, and in order to show how these activities apply to mining and other mineral industries, the discussions and descriptions given in this bulletin are grouped mostly according to the investigations carried on by the several divisions of the bureau, as follows: Mining, fuels and mechanical equipment, mineral technology, petroleum technology, and metallurgy.

MINING INVESTIGATIONS.

Under the general title of mining investigations are included accounts of the educational and life-saving work done through the mine safety cars and stations; field investigations of coal and metal mines, with respect to the health and safety of miners and the efficiency of the mining methods in reducing or preventing waste of mineral resources; and the Government inspection of mines on Indian lands and of mines in the Territory of Alaska.

MINE-SAFETY INVESTIGATION.

NEED OF SAFETY MEASURES.

To show the need that has existed for energetic efforts to reduce the loss of life at mines in the United States, a brief summary of the fatalities in coal mining will suffice.

The United States has produced 9,838,300,000 tons of coal under inspection since the first inspection law was enacted in 1870. Complete records for this production show that 54,453 men have been killed by accidents while mining coal.

The number of men engaged in the production of this amount of coal represents the equivalent of an army of 16,500,000 men engaged for 1 year. The number of men killed per 1,000 employed during the 46-year period was 3.30. For every 10,000,000 tons of coal produced 55 lives were lost, or a production of 180,676 tons for each fatality.

That the accident rate is being reduced is evidenced by the fact that the fatality rate in 1915 per 1,000 men employed was 2.95, which was the lowest recorded since 1898. The production of coal per fatality in 1915 was 228,600 tons, which was the largest production per fatality during the history of coal mining in the United States. The number of men at present enrolled in the coal-mining industry is approximately 765,000 a year.^a

The conservation of human life in mines and industrial plants is of paramount importance and is being considered by legislative bodies, labor organizations, captains of industry, and individual employees. Every fatal accident leaves its impress on the community in the loss of a useful citizen and the provider for a family, with the result that many widows and orphans are rendered public charges for which taxpayers must contribute support. The sufferings and

^a Fay, A. H., *Coal-mine fatalities in the United States, 1870 to 1914, with statistics of coal production, labor, and mining methods, by States and calendar years*: Bull. 115, Bureau of Mines, 1916, 370 pp., 3 pls., 13 figs.

privations borne by many of the dependents can not be measured by words nor compensated by a money equivalent.

Here is an industry employing 765,000 men, of whom 3 out of every 1,000 are killed each year, and at least 180 per 1,000 are injured to the extent that time is lost and in most cases medical attendance requested. A reduction of 50 per cent in the number of fatalities would result in an annual saving of fully 1,200 lives, to say nothing of the reduction in injuries and sufferings now sustained by nearly 150,000 unfortunates. From the point of view of the humanitarian, no greater good for the industry can be accomplished than to effect this reduction, and it is to this end that the Bureau of Mines is conducting its campaign of education and safety first. Experience is showing that accidents may be reduced in number, although they can not be eliminated entirely.

EDUCATIONAL FEATURES.

No method adopted by the bureau for the purpose of interesting the miner in mine-safety work has been received with greater enthusiasm or more widespread appreciation than that of illustrated lectures. The giving of such lectures is part of the duties of every field man of the bureau.

So great has been the demand from both operators and miners for these lectures that, because of lack of sufficient funds, the bureau has not been able to meet it fully. Both lantern slides and motion pictures are used to illustrate different phases of coal and metal mining, sanitary and welfare work in mining communities, rescue and recovery work, and first-aid demonstrations.

Over 43,000 persons attended the lectures of the bureau's field men during the fiscal year ended June 30, 1916, and it is felt that the deep interest manifested by both the miner and operator is evidence that much good has resulted.

This work is only in its infancy, and as the field is broad it will probably demand more attention each year.

MINE SAFETY CARS AND STATIONS.

For the purpose of training miners in first aid to the injured and in the use and care of mine rescue apparatus and for facilitating investigations of mine disasters the bureau maintains five stations and eight mine safety cars in the coal and metal-mining regions of the United States. The headquarters of the different cars are situated as follows: Huntington, W. Va.; Evansville, Ind.; Ironwood, Mich.; Pittsburgh, Kans.; Butte, Mont.; Reno, Nev.; Raton, N. Mex.; and Pittsburgh, Pa.

The mine safety stations are at Pittsburgh, Pa.; Jellico, Tenn.; Seattle, Wash.; McAlester, Okla.; and Birmingham, Ala. Each sta-

tion is in charge of a foreman miner, who gives both mine rescue and first-aid training.

The mine safety cars, some of which were Pullman sleeping cars (see Pl. I, A) and some of which were specially built for mine safety service, now have accommodations for about 12 persons besides apparatus for use after mine disasters and for first-aid and rescue training. The cars are transported free of cost to the Government, grateful acknowledgment being herewith made to the following companies for their cooperation in this regard:

Railroads with which the Bureau of Mines has special arrangements for handling mine safety cars.

[In effect during 1916.]

Alabama Great Southern Railroad.
 Arizona & New Mexico Railway.
 Arizona Eastern Railroad.
 Arkansas Central Railroad.
 Atchison, Topeka & Santa Fe Railway.
 Baltimore & Ohio Railroad.
 Bellefonte Central Railroad.
 Bessemer & Lake Erie Railroad.
 Buffalo, Rochester & Pittsburgh Railway.
 Buffalo & Susquehanna Railway.
 Butte, Anaconda & Pacific Railway.
 Canon City & Cripple Creek Railroad.
 Carolina, Clinchfield & Ohio Railway.
 Central of Georgia Railway.
 Central Railroad of New Jersey.
 Chesapeake & Ohio Railway.
 Chicago, Burlington & Quincy Railroad.
 Chicago & Eastern Illinois Railroad.
 Chicago & North Western Railway.
 Chicago Great Western Railroad.
 Chicago, Indianapolis & Louisville Railway.
 Chicago, Milwaukee & Puget Sound Railway.
 Chicago, Milwaukee & St. Paul Railway.
 Chicago, Rock Island & Pacific Railway.
 Chicago, Terre Haute & Southeastern Railway.
 Cincinnati, New Orleans & Texas Pacific Railway.
 Cincinnati Southern Railroad.
 Coal & Coke Railway.
 Colorado Midland Railway.
 Colorado & Southeastern Railway.
 Colorado & Southern Railway.
 Colorado Springs & Cripple Creek District Railway.
 Colorado & Wyoming Railway.
 Columbia & Puget Sound Railroad.
 Cripple Creek & Colorado Springs Railroad.
 Crystal River Railroad.
 Cumberland & Pennsylvania Railroad.
 Delaware & Hudson Railroad.
 Delaware, Lackawanna & Western Railroad.
 Denver & Rio Grande Railroad.
 Detroit, Toledo & Ironton Railroad.
 Duluth & Iron Range Railroad.
 Duluth, Missabe & Northern Railroad.
 Duluth, South Shore & Atlantic Railway.
 El Paso & Southwestern Railroad.
 Erie Railroad.

Evansville & Terre Haute Railroad.
Florence & Cripple Creek Railroad.
Fort Smith, Subiaco & Eastern Railroad.
Fort Smith & Western Railroad.
Golden Circle Railroad.
Great Northern Railway.
Hocking Valley Railway.
Illinois Central Railroad.
Kanawha & Michigan Railway.
Kansas City, Mexico & Orient Railroad.
Kansas City Southern Railway.
Lackawanna & Wyoming Valley Railroad.
Lake Superior & Ishpeming Railway.
Lehigh Valley Railroad.
Louisville, Henderson & St. Louis Railway.
Louisville & Nashville Railroad.
Midland Terminal Railroad.
Midland Valley Railroad.
Mineral Range Railroad.
Minneapolis, St. Paul & Sault Ste. Marie Railway.
Mississippi River & Bonne Terre Railway.
Missouri, Kansas & Texas Railway.
Missouri & North Arkansas Railroad.
Missouri, Oklahoma & Gulf Railway.
Missouri Pacific Railway.
Mobile & Ohio Railroad.
Monongahela Railroad.
Montana, Wyoming & Southern Railroad.
Munising, Marquette & Southeastern Railway.
Nashville, Chattanooga & St. Louis Railway.
New York Central Lines.
New York, Ontario & Western Railway.
Norfolk & Western Railway.
Northern Pacific Railway.
Oregon Short Line.
Oregon-Washington Railroad & Navigation Co.
Pennsylvania Railroad.
Philadelphia & Reading Railway.
Pittsburgh, Shawmut & Northern Railroad.
Queen & Crescent Route.
Quincy, Omaha & Kansas City Railroad.
Ray & Gila Valley Railroad.
Salt Lake Route.
San Pedro, Los Angeles & Salt Lake Railroad.
Santa Fe, Raton & Eastern Railroad.
Southern Indiana Railway.
Southern Pacific Railroad.
Southern Railway.
Southern Railway in Mississippi.
St. Louis Southwestern Railway.
St. Louis & San Francisco Railroad.
St. Louis, El Reno & Western Railway.
St. Louis, Rocky Mountain & Pacific Railway.
Tacoma Eastern Railroad.
Tennessee Central Railroad.
Texas, St. Louis & Western Railroad.
Toledo, Peoria & Western Railway.
Toledo, St. Louis & Western Railroad.
Union Pacific Railroad.
United Verde & Pacific Railway.
The Virginian Railroad Co.
Virginia & Truckee Railway.
Wabash Railroad.
Western Maryland Railway.
Western Pacific Railway.
Wheeling & Lake Erie Railroad.
Wichita Falls Route.



A. INTERIOR OF BUREAU OF MINES MINE-SAFETY CAR.



B. MINE RESCUE CREW UNDERGROUND DURING RESCUE MANEUVERS AT VANDALIA NO. 20 MINE, LINTON, IND.

Each car when fully manned has a foreman miner to give training in modern safety and rescue methods, a first-aid miner to give training in first aid to the injured, and a district mining engineer and a surgeon to confer with mine operators, mine physicians, and others in their investigative work.

The cars are in charge of practical men, and when funds are available for their operation they move about through the various fields, arousing interest in the movement for greater safety in mining.

In addition to the cars and stations, three motor trucks have been added. The trucks are each equipped with 10 sets of rescue apparatus and supplies sufficient for 48 hours of continuous apparatus work. One is stationed at Pittsburgh, Pa., one at Birmingham, Ala., and one at Seattle, Wash. They are of special value for giving aid after accidents in mines within a radius of 40 or 50 miles of their headquarters, as they can give efficient service on short notice, carrying to a disaster a rescue crew of 10 men ready for duty.

MINE RESCUE AND FIRST-AID TRAINING WORK DURING ONE YEAR.

During the period in which the crews of the bureau's mine safety cars and stations conducted active training in the fiscal year ended June 30, 1916, 62,693 miners visited the cars and stations, 43,060 attended lectures and safety demonstrations, 285 received mine rescue training, 5,598 were given first-aid instruction, and 2,610 received both first-aid and mine rescue training, the total number trained being 8,493, which is an increase of 494 over the previous year. A still better showing would have been made had it not been for the fact that many of the cars were in the shops for repairs for a part of the time.

The training is well received throughout the country; in fact, in Pennsylvania there have been numerous calls for mine rescue instruction which the bureau has not been able to satisfy, owing to its meager force.

It might be mentioned that the casualty-insurance companies have recognized the value of the bureau's first-aid and rescue training by making reductions in their rates for any mine that has a certain number of men trained by the bureau. This action has stimulated the desire for training and will undoubtedly further stimulate it as compensation laws and mine insurance become more general.

The individual miner has recognized the value of the first-aid work not only to his fellow miners but also to himself, for he realizes that a knowledge of first aid may some day enable him to save his own life or the life of some member of his immediate family through the prompt control of arterial bleeding or the protection of wounds

to prevent infection. It is perhaps largely because the training offers something he considers of personal benefit that the miner has taken so enthusiastically to the work, and in many fields the instructors have had classes of 50 to 100 men at night. Ordinarily these men take the training at night on their own time and after they have done a hard day's work in the mines. In addition, many walk 3 or 4 miles from their homes to the cars at which the instruction is given.

During the past fiscal year, in many local, State, and district first-aid meets, the bureau's men assisted in training competing teams and in organizing the meets.

At each point visited by the bureau's mine safety cars a first-aid society was organized; these are making the training of a more lasting value to the community by insuring a continuance of the instruction.

Frequently rescue maneuvers are conducted underground after a class has been trained in the use of rescue apparatus. These maneuvers are conducted under conditions representing as nearly as possible those likely to be encountered in actual rescue and recovery work. Plate I, *B*, shows a crew participating in underground maneuvers.

FIELD CONTESTS IN MINE RESCUE AND FIRST-AID METHODS.

Forty-seven mine rescue and first-aid contests and field meets were held during the fiscal year 1916 under the auspices or with the assistance of the Bureau of Mines. A few of these contests were company affairs, but most were intercompany or interstate. In many contests the Bureau of Mines employees arranged and supervised the events, and in nearly all, at the request of the operators and miners, gave the competing teams special instruction and training. Plate II, *A*, shows a first-aid team that has completed caring for a "wounded" man in a first-aid contest.

These contests were held at points widely distributed throughout the United States from Alabama to Montana, demonstrating the interest and enthusiasm of miners and mine officials in carrying forward mine safety work and in furthering the great movement for greater safety in mining.

ACCIDENTS INVESTIGATED DURING 1916.

In the 89 accidents investigated by Bureau of Mines employees during the fiscal year 1916 there were 285 killed, 3,015 escaped unassisted, and 75 were rescued through the efforts of volunteer miners, company officials, State mine inspectors, and company rescue crews.



A. A COMPETING TEAM IN FIRST-AID WORK.



B. RESCUE CREW ENTERING MINE.

Maneuvers at experimental mine at Bruceton, Pa.

Of the 89 accidents investigated, 68 were in coal mines, 14 in metal mines, 1 in a hydraulic pit, 1 in a quarry, and 1 in a strip pit; a dynamite explosion in the St. Louis sewer tunnel, a cave-in in a New York City subway, a boiler explosion, and an explosion in the new city sewer of Pittsburgh, Pa., were also investigated.

OPERATIONS AT MINE DISASTERS.

Besides giving training in mine rescue and first-aid work, the engineers, foremen miners, and first-aid miners of the mine safety cars and safety stations investigated numerous mine accidents and assisted in actual mine recovery (see Pl. II, B) and first-aid work after mine disasters.

Systematic methods of rescue and recovery work after mine fires and mine disasters are being improved each year, and it is felt that the increased efficiency of such methods has resulted largely from the unceasing efforts of the men trained by the Bureau of Mines or at State or privately owned rescue stations.

A fact worthy of emphasis is that during the fiscal year 1916 no explosion occurred in which a relatively large number of lives were lost. The Bureau of Mines, the mining departments of the various States, the operators, and the miners, through concerted and persistent endeavor, have brought about this gratifying result.

USE AND TESTING OF OXYGEN RESCUE APPARATUS.

The value of oxygen rescue apparatus is becoming more and more appreciated for both rescue and recovery work in mines. There has been a steady growth in the establishment of rescue stations throughout the country, particularly in the western coal fields.

In training miners the Bureau of Mines teaches the use of three types of apparatus—the Draeger, the Fleuss, and the Westfalia. The bureau itself is developing a fourth type, known as the Gibbs. Each of these types has a steel cylinder containing oxygen at a pressure of approximately 2,000 pounds to the inch and a reducing valve that allows a definite quantity of oxygen per minute, at a pressure slightly above atmospheric, to pass from the cylinder to a reservoir from which it is breathed by the wearer. The air exhaled by the wearer flows through to a compartment containing regenerating material, usually caustic soda (sodium hydroxide), by which the carbon dioxide in the breath is removed. The regenerated air joins the stream of oxygen from the reducing valve and is breathed again.

The possibilities and the limitations of such apparatus are becoming more thoroughly understood from year to year. However, during the fiscal year 1916 there were two instances of men wearing apparatus to combat mine fires with presumably little or no previous training in its use. In another instance an apparatus party of only two men made an exploration of considerable length, and one of them lost his life. Until such practices are discontinued the dangers incident to wearing apparatus in irrespirable atmospheres will not be reduced to the minimum.

Some men while wearing the apparatus fail to appreciate the fact that the weight of the apparatus causes them to become exhausted much more quickly than when working without it.

During the fiscal year 1916 tests were made with the four types of mine rescue apparatus mentioned to determine the accumulation of hydrogen and nitrogen in each type when the "oxygen" used was similar to that furnished the various rescue cars and stations—that is, oxygen mixed with nitrogen up to 3 per cent and hydrogen up to 1.3 per cent. In these tests 170 samples were taken of either the original oxygen or of the air breathed by the wearers, and the tests indicated that oxygen mixed with more than 0.2 per cent of hydrogen, or 2.5 per cent of nitrogen, might prove dangerous for use in mine rescue apparatus. Because of these tests the Bureau of Mines proposes in buying oxygen for use on its cars and stations to specify that the percentages of hydrogen and nitrogen must not exceed the figures mentioned. These tests also served to indicate the efficiency of the regenerator furnished with the Gibbs apparatus.

Because of the war in Europe the German company making the Draeger apparatus has been unable to procure regenerating cartridges from Germany for use in this country, and it has been necessary to have them manufactured in America. The Bureau of Mines has tested the efficiency of American-made regenerators. Two types were submitted, one containing granular sodium hydroxide and the other containing sheet sodium hydroxide. Generally speaking, all of these generators proved satisfactory, but those containing caustic soda in sheets and having absorbent blotters in the four upper trays showed the best results; they are now being used in the training work of the bureau.

Two tests were made with the Fleuss apparatus to determine whether caustic soda stored for some time in a Fleuss breathing bag was still in condition to efficiently absorb the carbon dioxide exhaled by the wearer. In one test caustic soda stored for one month in the breathing bag was tested; the caustic soda tried in a second test had been left in the bag for two months. Samples taken at the beginning and the end of the storage periods and eight inhalation-air samples

taken in the course of the two two-hour tests of each apparatus were analyzed. The tests showed that the sodium would still act as an efficient absorber, although the carbon dioxide in one sample of the inhaled air was as high as $1\frac{1}{2}$ per cent.

In trials with the Fleuss apparatus, electrolytic sodium hydroxide prepared in the lump form was tested. Eight inhalation-air samples taken in the tests showed the lump sodium to be an efficient absorption agent.

In four tests of the Fleuss apparatus the wearer tried the half mask, intended for use in fire fighting, instead of the mouthpiece. The tests indicated that although the half mask may seemingly have an air-tight fit on a man before he enters the smoke room, inward leakage may occur after he has been in a noxious or poisonous atmosphere for a short time; so that the half mask is not as safe as the mouthpiece.

TESTS AT PIKES PEAK.

An important series of tests of mine-rescue apparatus was made in Colorado to determine, if possible, whether rescue apparatus possessed certain defects that rendered it peculiarly dangerous for use at high altitudes. The first series of tests were made at Manitou, the second at the Half Way House on Pikes Peak, and the third at the summit of the peak. The results indicated that rescue apparatus is not more dangerous at high altitudes than at low. The oxygen supply was sufficient, and, in fact, men wearing the apparatus and getting a constant supply of oxygen from it could make physical exertions of which they were incapable when breathing the natural air because of its being rarefied and containing a less weight of oxygen per cubic foot than the air supplied by the apparatus.

The conclusion reached, therefore, was that the probable cause of the excessive number of accidents with rescue apparatus in mines at high altitudes resulted from neglect in the care of the apparatus, particularly in the renewal of rubber parts that had dried and cracked.

A secondary, but, possibly, more important result of these investigations was the comparison of the Gibbs apparatus with the older forms of apparatus, particularly the Fleuss and Draeger, heretofore used by the mine-rescue crews at the bureau. The results demonstrated conclusively that the Gibbs apparatus adapts itself to the wearer's needs far better than the older forms of apparatus. In endurance tests particularly it was found that this automatic adjustment tends to insure a much longer duration of the oxygen supply and of the power to absorb carbon dioxide than do any of the other types.

DEVELOPMENT OF NEW TYPE OF OXYGEN RESCUE APPARATUS.

As the Gibbs apparatus, which has been developed by W. E. Gibbs, engineer of mine-safety investigations, in cooperation with other members of the Bureau of Mines, seems in several respects to be superior to other types, a detailed description is here presented.

Rescue crews have to do hard work in the poisonous atmosphere of a mine after an explosion or fire, hence the breathing apparatus they wear must be of the best design and construction. It must be absolutely reliable, supply an artificial atmosphere of great purity, be as light as is consistent with strength, and impede the movements of the wearer as little as possible.

The Gibbs apparatus is a self-contained unit carried wholly on the back of the user. It is light and its parts are well protected against injury. A special device feeds the oxygen used, and although plenty is available for the wearer when working hard none is wasted when he is resting. Hence the new apparatus may be worn for a considerably longer time without recharging than can models now in use which furnish a constant volume. An unusually efficient carbon dioxide absorber that liberates little heat is another feature of the apparatus. Caustic soda, which is much cheaper than the potash salt formerly thought necessary, is used as the absorbent.

It is theoretically easy to supply a mine rescue man for two or three hours with an artificial atmosphere from an apparatus of no greater weight than he can readily carry on his back. In practice, however, the construction and operation of such devices offer many difficulties. The knowledge that the failure of any part to respond to the needs of the wearer means his almost certain death places a heavy responsibility on the designer and the constructor.

Normal air contains roughly 20 per cent of oxygen mixed with about 80 per cent of nitrogen and a trace of carbon dioxide. At each inspiration part of the oxygen breathed combines in the lungs with carbon brought by the blood, and the air expired contains about 4 per cent of carbon dioxide (CO_2). The nitrogen of the air is unchanged by the act of respiration and takes no active part in it other than to dilute the oxygen.

For all practical purposes an artificial atmosphere of pure oxygen, or oxygen containing only a small percentage of nitrogen, is actually preferable to normal air in the conditions surrounding mine rescue work. In spite of the general belief that pure oxygen is unsafe to breathe, no abnormal effects attend its use unless it be breathed for a much longer period than that during which rescue apparatus is customarily worn, and even then the only symptom noted is a slight irritation of the bronchial passages.

The amount of oxygen consumed in the body is precisely the same whether the gas is breathed pure or diluted with nitrogen in the form of air. Contrary to the belief held a few years ago, there is no flushing of the face, no feeling of exhilaration, no increase in the pulse rate, nor elevation of arterial tension.

If, however, the oxygen content of the air breathed be materially reduced, unconsciousness and death are almost sure to follow without any warning symptoms, provided the CO_2 content of the air remains low. For this reason it is advisable that breathing apparatus supply an atmosphere rich in oxygen. As much of the oxygen made from liquid air contains 2 or 3 per cent of nitrogen, which remains unchanged and accumulates in the apparatus, analyses of the atmosphere breathed by the wearer generally show a decreasing content proportionate to the length of time the apparatus is worn. If the proportion of carbon dioxide in the artificial atmosphere rises much above 2 per cent, deeper breathing or panting warns the wearer of danger, generally in time to let him get to safety.

Under the best conditions the exploration of mines in which there has been a recent fire or explosion is a hazardous undertaking, not only because of exposure to bodily injury but also because no mechanical breathing device can be made so perfect that it will never fail. Frequent inspection of the breathing apparatus and constant vigilance in keeping it in perfect repair reduce the risk of failure to the minimum.

The elements that enter into the construction of the Gibbs breathing apparatus are shown in figure 1. When the wearer inhales through the mouthpiece *a*, the valve *b* opens and oxygen passes from the bag *c* through the cooler *d* to the lungs.

On exhalation the oxygen, somewhat diminished in volume and containing about 4 per cent of CO_2 , issues from the mouth. The valve *b* now closes but the valve *e* opens to let the mixture of oxygen and carbon dioxide pass into the absorber *f*, where the caustic soda combines with the carbon dioxide to form sodium carbonate

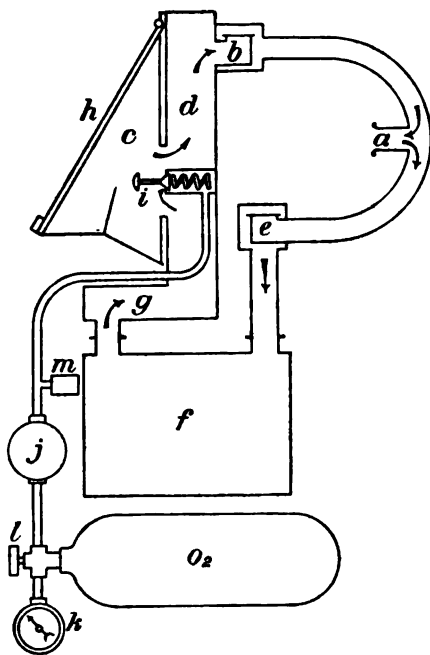


FIGURE 1.—Diagram showing circulation of respiratory air in Gibbs apparatus.

(Na_2CO_3), with the formation of some water and the liberation of heat.

From the absorber or regenerator the purified oxygen passes by way of the duct *g* to the breathing bag *d*, which expands to make room for it. The total volume of gas in the apparatus is now less than the original volume by the amount of carbon dioxide taken up by the absorbing can. After the wearer of the apparatus has taken a few breaths, the bag *c* collapses enough to permit the weighted lever *h* to open the oxygen-admission valve *i*, when the bag *c* fills again automatically.

The heat generated in the absorber is removed from the gas by radiation, partly from the cooler *d* and partly from the bag *c* and the connecting tubes.

A reducing valve *j* lowers the pressure of the oxygen from about 2,000 pounds per square inch in the bottle to a pressure that may be controlled by the admission valve *i*.

A pressure gage *k* indicates the available oxygen. The gas may be turned off by the stop valve *l* when the apparatus is not in use.

A relief valve *m* operates when the pressure in the circulatory system becomes too high.

In order to be practicable, breathing apparatus should be mounted on a suitable frame and be conveniently supported on the user; also, the various elements should be as nearly perfect in operation as mechanical ingenuity and good workmanship can make.

In breathing apparatus as previously constructed the flow of oxygen from the tank of compressed gas has been constant, not because of such a flow being desirable, but because of the difficulty of making a reducing valve that permits the flow to be interrupted.

With such a system the valve must be set to deliver oxygen at a rate that supplies the demands of the wearer when putting forth his utmost exertion. At all other times the supply is excessive and the surplus must be allowed to escape through a relief valve. Besides being wasteful of oxygen, such a system makes the pressure of the atmosphere within the apparatus fluctuate considerably and at times renders breathing difficult.

It was felt that the first step toward improvement should be to design a better reducing valve. A series of experiments led to the construction of a reducing valve that satisfies every requirement. A magnalium casing contains the valve-closing toggles, which are actuated by a flexible metallic bellows. Except for a rubber gasket at the junction of the casing and the toggles, all the parts are of metal. When this reducing valve is attached to a cylinder of gas at a pressure of 150 atmospheres, the pressure inside the bellows, with the outlet closed, is about one-sixth of an atmosphere. This remains constant even after the outlet has been shut several hours. Such a

reducing valve makes it possible to admit oxygen intermittently to the breathing bag in the exact quantity required by the user under conditions varying from complete rest to extreme labor.

All parts of the breathing apparatus are mounted on a frame of steel tubing which is carried on the back. Plate III, *A*, shows the apparatus in use with its protecting cover of aluminum removed.

The operation of the apparatus is as follows:

On inhalation, air from the breathing bag lifts the inhalation valve *b* (fig. 1) and passes by way of the flexible tube and mouthpiece to the lungs. Exhaled breath passes to the exhalation valve *e*, thence down a flue to the absorber *f*, where it loses its CO₂, thence up through the cooling can *d*, where it loses heat, into the breathing bag *c*.

The absorbing can *f* has been the subject of much experiment. A form tentatively adopted contained 20 vertical sheets of fine iron-wire gauze, held parallel to each other and one-fifth of an inch apart by spacers. These sheets before being put in the can were dipped in molten caustic soda (NaOH) containing 20 per cent of water. The caustic solidified on the gauze when cold, forming reinforced plates about a sixteenth of an inch thick, between which the expired air passed.

The plates not only absorb carbon dioxide from the wearer's breath but maintain a uniform surface from which the condensed and chemically produced moisture drains away, so that the capacity of the absorber is nearly constant until the active material has all been used. Recently another type of absorber, containing caustic soda in lumps, has been adopted.

A pressure gage or "finimeter," which is read by touch, instead of sight, and sounds an alarm when the oxygen in the cylinder has been reduced to 30 atmospheres, completes the apparatus.

The whole device, which weighs only 30 pounds, or considerably less than other types, is suspended from the shoulders by leather straps. There is a minimum of parts. All the connections have been made without the use of rubber wherever possible.

A simple mouthpiece and nose clip is used instead of a helmet. Experience has shown that the helmet is dangerous, and its use has been abandoned by the Bureau of Mines.

The pressure within the apparatus is maintained slightly above that of the atmosphere by the pressure of a weighted flap, *h* (fig. 1), on the breathing bag. Consequently, if the apparatus is punctured, or if a crevice in any part of the system opens, the leakage is outward only.

Exhaustive tests of the new apparatus have shown that it permits unusually free breathing; that the air supplied is comfortably cool; that as the front of the body of the wearer is entirely free he is not

hampered in his movements; and that the parts of the device are well protected against accident. The entire apparatus can be quickly taken apart or put together with a wrench and screw driver.

STANDARDIZATION OF FIRST-AID METHODS.

In January, 1916, with a view to standardizing first-aid practices and dressings, the bureau called into conference its staff of consulting surgeons, its mine surgeon, and a representative of the American National Red Cross. The conference decided that the bureau's first-aid work ought to aim at the following essentials: Efficiency; simple materials for dressing; simple methods of applying dressings; and consideration of cost of material recommended for dressings.

All existing practices and dressings were considered and those that best complied with the essentials named were chosen.

The methods and dressings recommended by the staff are now being taught by all the bureau's instructors, and bureau publications describing them are being published.

In the past there has been much confusion at first-aid contests, owing to dissimilar dressings being applied. It is hoped that the new standards will go far toward eliminating such confusion.

A number of contests in first aid have already been held under these standards, the first being at Clinton, Ind., on April 1, 1916, where it was evident that the standardization work had already resulted in materially reducing the confusion mentioned.

Suggestions were offered by the standardization committee as follows:

1. That, as far as possible, first-aid training be given under the immediate supervision of a regularly registered and qualified physician.
2. That there be close cooperation with the first-aid department of the American National Red Cross in first-aid work.
3. That all examinations for first-aid certificates shall be held by a qualified physician and shall conform with such standards as may be laid down by the Bureau of Mines.
4. That it shall be the imperative rule in all first-aid contests that the judges shall be regularly qualified physicians familiar with first-aid work.
5. That where it is possible every employee in a mine should be trained in first aid; and if this is impossible, that at least one out of every ten employees, both inside and out, should receive such training.



A. GIBBS OXYGEN MINE RESCUE APPARATUS, WITH COVER REMOVED.



B. TYPICAL APPEARANCE OF ROCK-DUSTED ENTRY IN A COAL MINE.

Note whitened effect on walls.



COOPERATIVE RESCUE AND FIRST-AID TRAINING.

To extend the usefulness of the bureau's work in training miners in rescue and first-aid work a plan for cooperating with privately or State-owned rescue stations has been evolved.

The bureau announced that it would list such stations as cooperative provided their equipment and facilities were found to meet the purpose of training. When a station asks to be placed on the list of cooperating stations a representative of the bureau is assigned to investigate the station and to report on its status. His recommendation is considered before the station is designated as cooperative.

Every cooperative station is allowed to train men so as to qualify them to receive Bureau of Mines certificates for mine rescue and first-aid work if it conducts training according to a schedule similar to that used by the bureau. At the conclusion of the training a formal application is made for an examination and further test of the fitness of the applicants to receive certificates.

When a class is ready for examination the bureau sends to the station one of its engineers or foreman miners, who conducts the examination and directs practical demonstrations in first-aid and mine rescue work. If the records of the examination and work are satisfactory to the bureau, certificates are issued. This cooperation, which is conducted with the least possible cost to the bureau, affords a greater number of miners an opportunity to receive training.

Under this plan cooperative training stations have been listed by the bureau as follows:

Arizona Copper Co.....	Morenci, Ariz.
Colorado Fuel & Iron Co.....	Jansen, Colo.
Copper Queen Consolidated Mining Co.....	Bisbee, Ariz.
Colorado School of Mines.....	Golden, Colo.
Detroit Copper Mining Co.....	Morenci, Ariz.
Ellsworth Collieries Co.....	Ellsworth, Pa.
Knox Mining Co.....	Jellico, Tenn.
Knox Mining Co.....	Rockwood, Tenn.
Missouri School of Mines and Metallurgy.....	Rolla, Mo.
Oliver Iron Mining Co.....	Ely, Minn.
Oliver Iron Mining Co.....	Eveleth, Minn.
Oliver Iron Mining Co.....	Hibbing, Minn.
Oliver Iron Mining Co.....	Iron Mountain, Mich.
Oliver Iron Mining Co.....	Ironwood, Mich.
Oliver Iron Mining Co.....	Ishpeming, Mich.
Pennsylvania State College.....	State College, Pa.
Ray Consolidated Copper Co.....	Ray, Ariz.
Republic Iron & Steel Co.....	Republic, Pa.
Superior Coal Co.....	Superior, Wyo.
Union Pacific Coal Co.....	Cumberland, Wyo.

Union Pacific Coal Co.....	Hanna, Wyo.
Union Pacific Coal Co.....	Rock Springs, Wyo.
United States Fuel Co.....	Salt Lake City, Utah.
United Verde Copper Co.....	Jerome, Ariz.
Victor American Fuel Co.....	Gibson, N. Mex.
West Virginia University.....	Morgantown, W. Va.

FIRST AID IN INDIANA.

The following resolutions* recommending first-aid training, the study of safety first, and participation in first-aid meets were adopted by delegates of the United Mine Workers of America assembled for the twenty-fifth consecutive and second biennial convention of District No. 11 at Evansville, Ind., on March 27, 1916. The resolutions were probably the first of the kind adopted by that organization:

Whereas the United Mine Workers of America believes in and practices fraternalism as exemplified by the motto, "United we stand, divided we fall"; and

Whereas it is the obligation of every member of our organization to aid and assist a brother in distress; and

Whereas there frequently occur times and opportunities to render such aid and assistance while engaged in the pursuit of our duties; and

Whereas for the benefit of injured mine employees the United States Bureau of Mines has introduced in the mining industry the movement known as first aid to the injured; and

Whereas this movement has proven its valuable and lasting worth by preventing suffering, saving lives, and giving our brothers reliable and efficient first-aid treatment; and

Whereas the United States Bureau of Mines introduced said first-aid movement and has at the cost of time and money been the most potent factor in teaching first aid to the injured: Now, therefore, be it

Resolved, That District No. 11 of the United Mine Workers of America, now in regular session, give the said first-aid movement our most hearty approval and indorsement; and be it further

Resolved, That each member of this organization, as well as the official body, give it their conscientious and whole-hearted support and assistance; and be it further

Resolved, That each member of this organization prepare and fit himself by a course of proper and efficient training in first aid, so that intelligent and efficient treatment may be rendered when a brother is injured; and be it also further

Resolved, That this convention indorse and approve the work of the United States Bureau of Mines in said movement; and be it further

Resolved, That the annual State first-aid contests held in Indiana hereby receive the approval, indorsement, and support of District No. 11, United Mine Workers of America; and be it further

Resolved, That each member of the organization be a constant student and practitioner of safety first.

* Proceedings of the twenty-fifth consecutive and second biennial convention of District No. 11, United Mine Workers of America, 1916, pp. 387-388.

COAL-MINING INVESTIGATIONS.**PREVENTION OF COAL-DUST EXPLOSIONS BY WATERING OR USE OF ROCK DUST.**

Widespread coal-mine explosions start from an ignition of gas or of coal dust or of a mixture of the two. Prevention of such explosions can be accomplished by removing the sources of ignition, or by removing or rendering inert any material that may be ignited. The safest plan is to attempt to do both.

Ignition of gas and dust is possible from open lights, sparks, arcs, or flames from explosives. The use of oil or gasoline permissible safety lamps or miners' permissible electric safety lamps instead of the common oil lamps, the removal of electrical apparatus from places where ignition may be caused, or safeguarding such apparatus so as to prevent its causing ignition, and the use of short-flame permissible explosives to blast down coal that has been properly mined or sheared are precautions that eliminate largely the possibility of an ignition.

Gas ignition can be prevented by providing ventilation adequate to dilute gas at all points where it is generated and adequate to prevent accumulations of gas. In any mine in which open lights are used, it is desirable that the proportion of gas in the returns should not exceed 0.25 per cent. In the returns of any mine in which permissible lights are used, sufficient volume of air should be introduced so the proportion does not exceed 1 per cent. Ventilation should be so controlled that not more than 2 per cent of gas can be detected at any point in the mine.

In attempts to prevent ignition of coal dust, effort is made to prevent as far as possible accumulations of coal dust, and by wetting the dust to render inert coal-dust accumulations that do occur so that the dust can not be blown into a cloud, or by adding incombustible matter enough to render the cloud nonexplosive.

Coal-dust accumulations can largely be avoided by replacing all shooting off the solid by shooting after the coal has been mined or sheared, using tight cars, loading cars so that the coal will not be spilled over the tops, and frequently cleaning the roads.

There are several ways of rendering coal dust inert by wetting. The air current can be humidified with exhaust steam or sprays either after being preheated by passing over and through radiators, or without preheating. If enough steam is added to saturate the air current at the mine temperature there will be no evaporation of water in the mine. However, in winter it is difficult to keep the current saturated and it is usually necessary to supplement humidifying with some watering.

Watering is used in many mines where humidifying the air current is not attempted. A water car or hose from pipe lines may be used. Water cars are of several types. In one type the water merely dribbles on the middle of the road; in another the water is under pressure and a strong spray is thrown on all surfaces of the entry as the car passes. The use of hose to wash down all surfaces thoroughly is the best method if done frequently enough. In winter it may be necessary to sprinkle throughout the mine every day.

Deliquescent salts, usually common rock salt and calcium chloride, are also used for wetting dust. The former is slightly deliquescent, and does no good except when the air is nearly saturated; then the salt becomes moist and helps to wet the dust. Calcium chloride is more efficient; applied in quantities of about 2 pounds per foot it will keep a roadway moistened for two or three months, depending on how fast the coal dust accumulates. However, this method does not care for dust on ribs, roof, and timbers.

The dry method of rendering coal dust inert, termed rock dusting, has been used only during the past eight years. Applying dry incombustible dust to the ribs, roof, and floor of entries, air courses, and rooms, after they have been cleaned as thoroughly as possible of coal dust, affords efficient protection for a long period, until enough coal dust accumulates to make the mixture of rock dust and coal dust explosive.

Atkinson in 1880 and Garforth in 1886 noted in England, and Rice in 1892 in the United States, that after certain mine explosions the tram roads on which there was much incombustible matter showed hardly any explosion effects. It was noted in Germany that fine sand spread over coal dust appeared to prevent the spread of flame from a blow-out shot. Although these facts were recorded at the time, the idea of using incombustible dust to prevent coal-dust explosions developed slowly. For this development Mr. Garforth was largely instrumental. In 1908, in the experimental gallery of the British Government at Altofts, England, a stone-dust zone 300 feet long extinguished the flame from a 170-foot zone containing much coal dust. Seemingly this was the first occasion on which rock dust was used to check explosions.

At about the same time in France Mr. Taffanel began experimenting along the same lines. In 1911 the United States Bureau of Mines began similar testing on a large scale and for the first time this was done in a real mine.

In 1912 a British commission that was in charge of the Altofts experiments issued a preliminary report recommending the use of rock dust. In their report they mentioned five British collieries that were then trying rock dust; since then through official approval

of the method many mines have adopted the application of rock dust as a safety measure.

In the United States rock-dusting on a large scale was first tried in a Colorado mine, where sprinkling had caused roof falls. The application of dry adobe dust began in March, 1911, and has been continued to date. The first dusting was by hand, as a blower does not give a satisfactory coating when the dust is first applied; for subsequent treatment a rock-dust blower was used. About 10 miles of entry has been kept dusted now for about four years, at a cost of about 0.3 cent per ton of coal produced. The dust is obtained from the country roads in a near-by canyon. It costs \$1 per ton, and about 2,500 tons have been used. It all passes through a one-eighth-inch screen and probably 20 to 30 per cent will pass through a 200-mesh screen.

The introduction of rock dusting at other American mines has been slow. In the meantime, by actual tests at its experimental mine the Bureau of Mines has been determining the proportion of incombustible that, when mixed with coal dust from the Pittsburgh bed, will make a nonexplosive mixture. The bureau has determined that an explosion once started will not propagate through a mixture containing 64 per cent of incombustible material if not more than 20 per cent of the dust will pass through a 200-mesh sieve, and if gas is absent from the air current.

During the latter part of the year 1914 one of the large coal-mining companies working the Pittsburgh bed agreed to cooperate with the Bureau of Mines in a test of rock dusting, and certain localities in one of its old drift mines that produces 1,000 to 1,500 tons of coal daily were selected for test purposes. The mine is in the Pittsburgh bed, is naturally dry and dusty, and produces some gas. Calcium chloride, salt, and water are employed to lay dust; electric lamps and permissible explosives are used entirely.

A carload of limestone dust was purchased as most available for use. For the localities to be dusted first, short zones of widely differing character were selected. One zone, about 450 feet long, was in the main haulageway about 2 miles from the opening and included a switching point at which there was much bumping of cars and spillage of coal. Another, about 560 feet long, was in a nearly level butt entry; a third, about 1,000 feet long, was in part on a hill where there was considerable spillage and the mules dug up the roadway considerably. These zones were along roadways ballasted mostly with coal dust and coal. The loose coal and rock were cleaned up before rock dust was applied. In a fourth zone rock dust was applied to a roadway that had been ballasted with roof shale. Other entries and butt entries dusted later differed from those already dusted only in that two of them at one time had cal-

cium chloride applied which was still evident in the greater dampness of the roadways.

The first two applications were made to these zones in January, February, and March, 1915. From 4 to 6 pounds of rock dust per linear foot of entry was applied by hand, being thrown on the ribs and roof so as to give a uniform white coating. That which did not stick fell to the floor and there mixed with the coal in the road dust. At intervals, generally one to three months, these zones have been redusted and extensions to other parts of the mine have been made so that in July, 1916, approximately 6,700 feet had been dusted.

About twice a month samples have been taken in the dusted zones with a specially designed scoop. Loose road material to a depth of about 1 inch is brushed into the sampling scoop from a section 6 inches wide, extending completely across the roadway. The scoop contains a 10-mesh screen. Material not passing through this screen is rejected; the undersize is taken for analysis. Dust brushed from a 6-inch wide section of the ribs and roof makes up the rib sample. The road sample and the rib sample considered together represent a complete cross section of the mine entry. In the laboratory all material in these samples that does not pass through a 20-mesh screen is rejected; the undersize is analyzed for moisture, ash, and carbon dioxide, the weights of the samples are recorded, and sizing tests are made with 48, 100, and 200 mesh screens. The sum of the moisture, ash, and carbon dioxide contents gives the amount of incombustible present. From relative weights and analyses the average incombustible content of ribs and floor is calculated; the carbon dioxide content gives an approximate idea of the amount of limestone present; and the sizing tests give a basis of comparison with tests at the bureau's experimental mine in which mixtures of sized material are used.

For the most part the samples taken have shown conditions to be satisfactory; for example, the average of 69 samples of rib and road dust sections showed a content of incombustible material of 65.2 per cent. It is considered that samples of rib and road dust taken together should show a content of incombustible material of at least 60 per cent when the proportion of 200-mesh coal dust present is not more than 20 per cent of the total coal dust present. Even when the samples fell below the standard given, they indicated conditions much better than in zones in which the wet method of rendering coal dust inert was used. The appearance of a rock-dusted area in an operating mine is shown in Plate III, *B*.

Limestone dust laid down at the mine cost \$2.90 per ton. The labor rate was \$2.62 per man per eight-hour day. At these figures, the cost of material and labor for the first application of 5.6 pounds of

limestone dust per linear foot of entry was \$1.76 per 100 linear feet. A second application of dust by hand at the rate of 2 pounds per foot cost 62 cents per 100 feet, which, on the basis of four hand applications per year, would make an annual cost of \$4.24 per 100 feet of entry. In January, 1916, the same coal company started dusting at two more of its mines. At one of these a little more than 3,000 feet of entry were dusted; in the other more than 7,500 feet.

Rock dusting at coal mines in this country has thus far been in the experimental stage. Costs can be greatly lowered. It is estimated that pulverized rock might be obtained for about \$1 per ton by using a small pulverizer at the mine to crush shale available there. Manufacturers are working on a machine that may do away with the need for making the first application of dust by hand; it has been shown that reapplying dust may be done better with a blower or similar machine than by hand. By using shale dust and reapplying it with a blower, it is estimated, on the basis of the costs given above, that \$130 per mile a year will safeguard a mine entry, or, roughly, for the mine in question, one-half cent per ton of coal mined.

Objection has been made to using rock dust on account of alleged danger to the miners' lungs. Pulverized shale dust has been shown to be noninjurious by British commissions which have particularly investigated this point. Dust containing free silica or sharp particles of any sort is likely to be injurious. Microscopic examination is a quick method of detecting the presence of sharp points or edges and the likelihood of danger from this source, and the Bureau of Mines has been making microscopic studies of various available materials for making rock dust.

On the whole, tests in this country as well as in Europe have sufficed to show the practicability and suitability of rock dusting as a means of preventing coal-mine explosions.

MINE AIR IN ILLINOIS COAL MINES.

The Bureau of Mines, in cooperation with the State of Illinois through the engineering experiment station of the University of Illinois and the State geological survey, is investigating the quality of the mine atmospheres in the coal mines of the State and the effect of different agencies on the air.

Under this general subject three publications have already been printed.* A short description of the results obtained in other studies of mine air, reports on which are in the course of preparation, follows.

* Darton, N. H., Occurrence of explosive gases in coal mines: Bull. 72, Bureau of Mines, 1914, 248 pp., 7 pls., 83 figs; Williams, R. Y., The humidity of mine air: Bull. 83, Bureau of Mines, 1914, 69 pp., 7 figs.; Williams, R. Y., Mine-ventilation stoppings, with especial reference to coal mines in Illinois: Bull. 99, Bureau of Mines, 1915, 80 pp., 4 pls., 4 figs.

STUDY OF RATE AT WHICH METHANE ACCUMULATES.

To obtain further information on the liability of methane accumulating in unsuspected places should the ventilation be temporarily shut off or interrupted, a number of samples were collected in Illinois and Indiana mines while the mine fan was stopped or while the ventilation was interrupted by blocking an air passage. Analysis of the samples showed that in some places where the gases diffused evenly when the fan was running at normal speed, the methane began to accumulate at the roof as soon as the ventilating current was stopped, and within a relatively short time the increase was enough to have greatly assisted in initiating a mine explosion and to have made a nonexplosive cloud of coal dust violently explosive. This investigation is of importance to those mines that practice slowing down the fan or otherwise interrupt the ventilating current before the firing of shots of black blasting powder or dynamite.

OCCURRENCE OF METHANE IN ILLINOIS COAL MINES.

The bureau's investigations made in cooperation with the State of Illinois have shown that there is a wide range in the amount of methane given off in the coal mines of that State. A large number of samples collected in two long-wall mines near La Salle were entirely free from methane, and the only samples containing methane from these mines showed less than 4 parts in 10,000 parts of air, or less than 0.04 per cent. On the other hand, some of the mines in Illinois may liberate large quantities of methane. One mine working the No. 5 bed in Saline County struck a zone of gas feeders which after 8 days gave off 40 cubic feet of methane per minute, or 57,600 cubic feet per day, from the face of two entries. Forty-eight days later this zone was giving off 24.7 cubic feet per minute. The coal had not been faulted or otherwise displaced but had been subjected to shearing stresses.

To prevent accidents from ignition of fire damp, the tendency in some Illinois mines has been to reduce the interval between the time the mine is examined and the time the men go to work in the morning. Also, more care is being taken to keep men from entering abandoned or caved workings. One method of doing this has been by the use of woven wire and poultry netting barriers with locked gates so that only authorized persons can pass through. (See Pl. IV, A.)

EFFECT OF USE OF EXPLOSIVES ON RETURN AIR.

To determine the effect of the gases from blasting on the mine air, a study was made of the return air current in an Illinois mine where approximately 18,000 cubic feet of air per minute was circulating.



A. METHOD USED IN ILLINOIS MINE TO FENCE OFF OLD WORKINGS WITHOUT INTERFERING WITH VENTILATION.



B. CANNON AND WIRING PREPARED FOR AN EXPLOSION TEST AT EXPERIMENTAL MINE.

After approximately 1,850 pounds of powder had been used for blasting, analyses of air samples showed that the change in the air was small, there being approximately 0.05 per cent increase in inflammable gases and 0.1 to 0.2 per cent increase in carbon dioxide, with a corresponding deficiency of oxygen. The slight increase of methane would indicate that little methane was liberated by the shooting. The return air was distinctly hazy, and the odor of sulphur made it unpleasant. The mine itself was generating a relatively small amount of methane.

ABSORPTION OF OXYGEN BY COAL AND ESCAPE OF CARBON DIOXIDE FROM IT.

A large number of samples of mine air were collected in Illinois mines to determine the ratio between the oxygen absorbed from the air by the coal and the amount of carbon dioxide added to the air. The ratio between oxygen absorption and carbon dioxide emanation was found to vary widely, but the average of a number of analyses appears to establish a ratio of about 3 to 1, or, in other words, while the air was in circulation about three volumes of oxygen were absorbed for one volume of carbon dioxide added.

WORK AT THE EXPERIMENTAL MINE.

CHARACTER OF MINE.

The experimental mine of the Bureau of Mines was opened in December, 1910, for the purpose of making explosion tests in connection with a study of the origin and progress of explosions of coal dust, of gas (methane), and of mixtures of coal dust and gas. The object was to develop and test methods of preventing or limiting such explosions. The mine is in the Pittsburgh coal bed and is situated on the side of a ravine near Bruceton, Pa., 13 miles from Pittsburgh. The Pittsburgh coal at this point is nearly level and is 5 to 5½ feet thick.

The mine layout as finally developed is shown in figure 2. It consists of two main entries 1,300 feet long; one pair of butts driven 300 and 350 feet to the left of the main entries, from which five rooms are turned; and a single butt entry 100 feet long, turned to the right from the main entry. The inner 350 feet of the main entries has a concrete lining. The purpose of this lining is to facilitate cleaning after explosion tests and to prevent particles of coal dust from the ribs and roof affecting tests with coals from other beds.

At intervals of 200 feet throughout the mine, except in the outer part of the main air course, large instrument stations of concrete are built in the rib. In these are placed measuring and recording instruments during explosion tests. In addition to these large sta-

tions are smaller intermediate stations. All the stations are connected to each other and to the outside by a cable in a pipe embedded in concrete in the rib. The mine has been fully described in Bulletin 56* of the bureau.

INSTRUMENTS USED.

The instruments used are of two general classes—pressure instruments, each of which records the explosion pressure at the point at which it is placed, and circuit breakers, each of which causes a break in the circuit when the flame of the explosion passes, this break being recorded by a chronograph in the observatory. In the latest type of pressure manometer the variation in pressure is re-

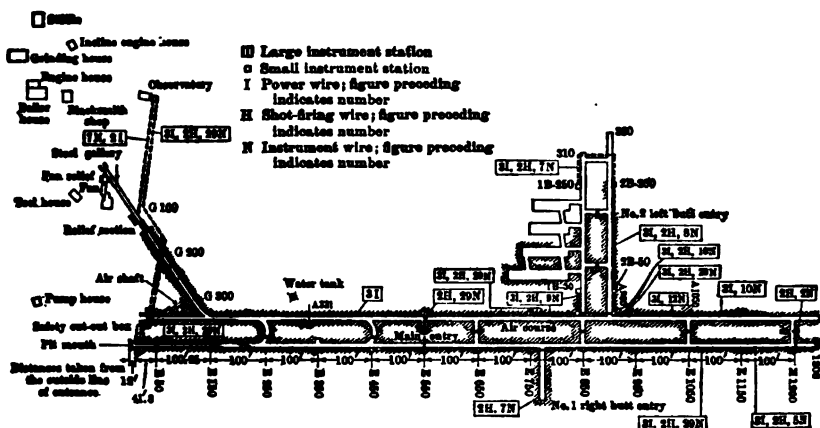


FIGURE 2.—Plan of experimental mine at Bruceton, Pa.

corded by a ray of light acting on a photographic film; the passage of the flame is also photographed on the same film. By means of the time records on the various films it is possible to plot the various records according to time and distance from the origin and to study the progress of the explosion as shown in the pressure and flame records. As the time the flame passes both large and small stations is indicated by the circuit breakers, the velocity between any of these points can be calculated. The principle of the breaking of circuits is also used in ascertaining the time of operation of various protective devices used in connection with the explosion tests.

PREPARATION AND DISTRIBUTION OF COAL DUST.

The coal dust used in the tests is prepared at a grinding house at the mine and immediately before a test is distributed along the entry on side shelves, on cross shelves about 8 inches beneath the roof, and

* Rice, G. S., and others, First series of coal-dust explosion tests in the experimental mine, 1913, 115 pp.

on the concrete floor, the usual distribution being at the rate of about 2 pounds of coal dust per linear foot of entry or 0.5 ounce per cubic foot of space. For standard testing an explosion is initiated by firing a blown-out shot of 4 pounds of FFF black blasting powder from a cannon placed on the floor of the main entry and pointed outward. Plate IV, *B*, shows the cannon and the wiring ready for an explosion test. The upper wires from the bore hole are the shot-firing wires, and the wire passing across in front of the hole is part of the ignition circuit; when the shot is fired this wire is broken, and the time of break, or of the starting of the explosion, is recorded in the observatory. The wires from the cannon pass through a pipe in the rib to the surface at the main opening. The igniting shot is fired electrically from the surface after everyone is out of the mine.

Plate V, *A*, shows the observatory instruments, called chronographs. Every pen is connected to some circuit, and when that circuit is broken the pen moves sidewise. The time is obtained from a record, made on each drum, of the oscillations of the tuning fork shown at the left of the chronograph.

When the shot is fired a flame about 20 feet long results. The air currents stir up the coal dust from the floor and the sides, causing a thick cloud, which is ignited by the flame. The inflammation of this coal-dust cloud sets up pressures which cause increased air movements and an extension of the cloud, which in turn ignites; this process continues indefinitely as long as there is fine, dry coal dust to be acted on. With pulverized Pittsburgh coal dust the explosion travels the first 150 feet in about 1 second; beyond this point the explosion increases in velocity and pressure until at a point 350 feet from the origin the velocity may be as great as 3,000 feet per second, with a pressure of 40 pounds per square inch. The extension of the explosion for 200 feet more may cause pressures as high as 100 to 120 pounds per square inch, which is about the maximum recorded by the instruments, although in all probability higher pressures have occurred in the explosion zone.

PURPOSES AND CHARACTER OF TESTS.

The first tests at the experimental mine were made to demonstrate to large numbers of visitors that coal dust would explode. Later, the systematic study of explosions was begun. Up to July 1, 1916, 330 tests had been made. In general the tests have been made in order to study these points:

The development of pure coal-dust explosions.

The relative explosibility of various coals.

The influence of the fineness and the quantity of the coal dust.

The influence of small percentages of gas in the air current.

The effect of the presence of rock dust.

The effectiveness of rock-dust barriers for limiting explosions.

The development of explosions in wide places.

In determining the explosibility of various coal dusts mixtures of coal dust and shale dust or other incombustible dust are used. The proportion of incombustible dust is increased until a mixture is found through which an explosion will not travel. The percentage of shale dust required to prevent the explosion is the measure of the relative explosibility of the coal dust.

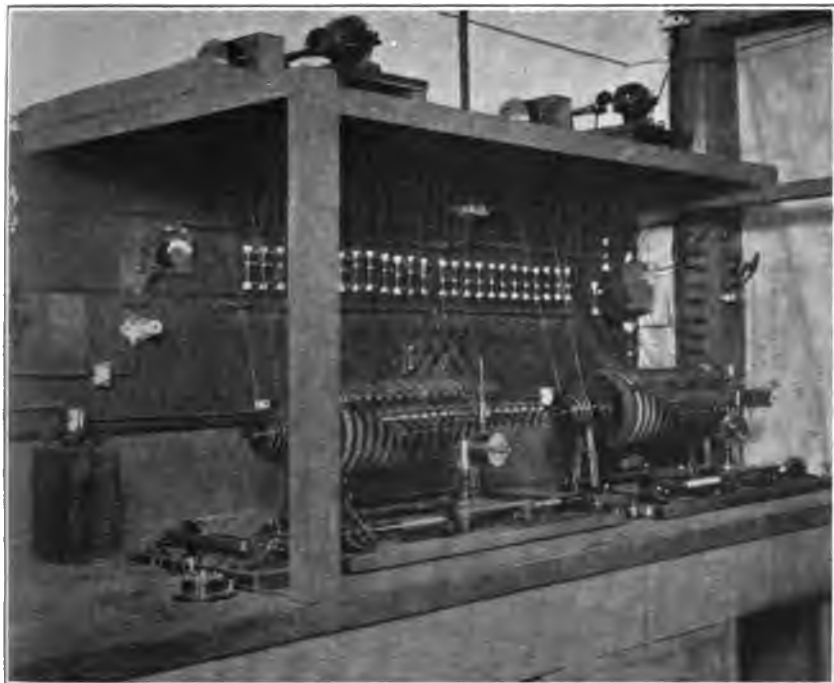
Two classes of tests, termed ignition and propagation tests, are made. In the ignition test the percentage of shale dust that prevents the mixture of shale and coal dust from being ignited by a standard blown-out shot from the cannon is determined; in the propagation test the percentage of shale dust that prevents propagation of an explosion started in a zone of more explosive dust, or by the ignition of a body of gas, is determined. After determination has been made of the ignition and propagation limits of coal dust in an atmosphere containing no explosive gas, tests are made when small percentages of gas are present in the air current, and the percentage of shale dust necessary, with various percentages of gas, to prevent ignition or propagation as the case may be is thus determined. Explosibility tests of this character have been made with 12 different coals, ranging from low-volatile anthracite to high-moisture lignites.

Another factor that affects the explosibility of coal dust is the fineness. A large number of tests have been made to determine the influence of this factor. With Pittsburgh coal dust the percentage of shale dust required increases from 50 to 80 per cent as the proportion of coal dust fine enough to pass a 200-mesh screen is increased from 10 per cent to 75 per cent. This shows how important is the fineness of the dust in a mine, for the finer the dust the greater must be the precautions taken to prevent widespread explosions.

METHODS OF MAKING COAL DUST NONEXPLOSIVE.

There are two general methods of rendering coal dust nonexplosive—first, by wetting the dust to prevent a cloud of dust from being formed, because only when the coal dust is in a cloud is it explosive; second, by adding to the coal dust enough incombustible dust to make the mixture nonexplosive. The first method is extensively used; the second is comparatively new in the United States. The advantages of using wet dust to prevent coal dust explosions have been pointed out in a preceding chapter.

The first coating is applied by hand and the use of limestone dust gives an entry the appearance of being whitewashed. Later when it



**A. CHRONOGRAPHS; INSTRUMENTS IN OBSERVATORY WHICH MEASURE SPEED OF EXPLOSIONS
IN EXPERIMENTAL MINE.**



B. BLOWING MACHINE FOR ROCK DUSTING IN COAL MINES.

is necessary to redust, the dust is blown into the air current by means of a blowing machine (Pl. V, *B*) and this dust settles down on ledges and surfaces on top of the coal dust that has accumulated, and keeps the latter from becoming explosive.

Ordinarily only the coal dust in the haulage entries is treated, but the parallel air course may contain considerable coal dust that would propagate an explosion. To obviate this danger, rock dust should also be blown into the air course.

The information gained from explosibility tests of the quantity of shale dust necessary to prevent mine explosions is very valuable in developing the rock-dusting method of rendering coal dust inert; if the explosibility of the road dust be questioned, analysis of samples will show whether the percentage of incombustible in the road dust is sufficient to prevent the propagation of an explosion along that road. Rough determination of the percentage of incombustible dust present can be made quickly with an instrument called a "volumeter," designed by Mr. Taffanel, of France.

DEVELOPMENT OF ROCK-DUST BARRIERS.

As has been indicated, in order to have efficient protection from widespread explosions, rock dusting should be done throughout a mine, as no one can foresee at just what point an explosion may start. As long as rock dusting or watering is satisfactorily maintained there should be no danger of an explosion being propagated throughout a mine; but if the method is used and is not adequately maintained there should be supplementary means of preventing an explosion from spreading throughout the mine and possibly causing the death of everyone in it. So-called "rock-dust barriers," based on devices tested in France, have been designed by George S. Rice, chief mining engineer of the Bureau of Mines, and tested at the experimental mine. Models of each of several types were developed which in a large number of tests effectively stopped the propagation of explosions.

One of the most easily constructed and effective of these barriers is known as the "trough barrier." It consists of six or more troughs extending across the entry at intervals of about 6 feet, with the bottom boards held in position by a lever system connected by wires to suspended vanes 100 feet on either side of the troughs. Should an explosion occur the vanes are caused to swing, thus releasing the bottom boards of the troughs so that the dust contents fall to the ground in a dense cloud. This dense dust cloud cools the gases to such a degree as to prevent propagation of flame beyond. Plate VI, 4, shows the appearance of a 6-trough barrier set in a mine entry.

The various types of barriers are described in Technical Paper 84* of the Bureau of Mines.

Such barriers are feasible at the entrances to all splits of the air current or sections of a mine, and at points along the main haulage roads, so that if an explosion occurs it will be arrested by the barriers, and will not spread throughout the mine. If an explosion should occur in a particular split in which the dust on the roads was explosive, it would not travel beyond the barrier at the entrance to the split. The men in the split might lose their lives, but the men in other parts of the mine would have a much better chance of getting out safely.

Most of the tests made have been in entries or narrow places, because most explosions are initiated in such places. However, as some explosions start in wide places, a few tests have been made with reference to the development of explosions in those places. These tests indicate that such explosions can be initiated with comparative ease by a single blown-out shot if enough fine coal dust be present. It was also indicated that cut-throughs (break-throughs) near the point of origin hindered the development of the explosion, as they permitted considerable release of pressure at the time when the explosion is just getting under way.

Plate VI, *B*, shows the effects of an explosion in the experimental mine.

LABORATORY STUDIES OF INFLAMMABILITY OF COAL DUST.

Large-scale tests of the explosibility of coal dust take time and are expensive, so that the desirability of a reliable laboratory test of inflammability is obvious. For some years the development of such a test has been in progress in the Bureau of Mines. The measure of inflammability now used in the bureau is the pressure developed when a definite amount of dust, air dried and ground until its particles are of definite size, is ignited by being blown in a definite way against a source of heat of definite size and temperature.

The apparatus used at present consists of a thick glass bulb connected through a tubulure with a steam-pressure gage, and another tubulure, opposite the first, which is connected air-tight with a small glass funnel whose stem is bent to a right angle. The dust to be examined is placed in this funnel. Between the two tubulures, in the center of the glass bulb, is a rigidly held platinum tube that is heated electrically to 1,200° C. The dust is blown against the platinum tube by a puff of oxygen under definite conditions. The

* Rice, G. S., and Jones, L. M., *Methods of preventing and limiting explosions in coal mines*: Tech. Paper 84, Bureau of Mines, 1915, 45 pp., 14 pls., 3 figs.



A. "SIX-TROUGH" ROCK-DUST BARRIER SET UP IN MINE ENTRY.



B. REINFORCED-CONCRETE STOPPING IN CROSS HEADING BLOWN OUT BY SUCCESSIVE DUST EXPLOSIONS IN EXPERIMENTAL MINE.

pressure developed by the ignition of the dust, which ranges from a few tenths of a pound to 20 pounds per square inch, is registered on the gage. Tests of mixtures of coal dusts and shale dust are also made.

In general, predictions from these laboratory tests of the explosibility of bituminous dusts in mine tests have been satisfactory, but there has not been such agreement between laboratory and mine tests of anthracite and lignite dusts. Further development of the apparatus is being attempted in order to obtain satisfactory results throughout the entire range of coals.

EFFECT OF MOMENTARY HEATING OF INFLAMMABLE COAL DUSTS.

Opinions differ as to how flame spreads through a cloud of inflammable coal dust. Some investigators believe that the dust particles burn as a whole, but others hold that during the short period of heating, just before inflammation, combustible gases are distilled from the coal, and that these gases give rise to the explosion. If the latter view is correct, then the readiness with which the gas distills is obviously a factor determining the inflammability or explosibility of the dust.

The plan of the bureau's experiments has been to subject a suspension of dust (dust cloud) to momentary heating sufficient to cause inflammation in air, to repeat the test in an inert atmosphere (nitrogen), and to analyze the atmosphere for combustible gases. The source of heat in some of the tests was an incandescent weight, which was dropped through the dust cloud. The results of such tests indicated practically no distillation of gases from the dust under the conditions of testing.

In work now under way the source of heat is a blast of hot gas. The apparatus used is shown in figure 3. Only the general features

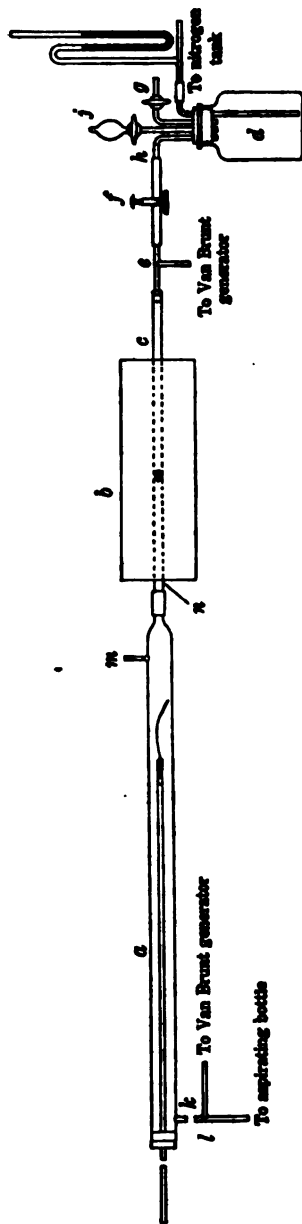


FIGURE 3.—Laboratory apparatus used in study of inflammability of coal dust.

need be described here. A glass rod passing through a rubber stopper in the wide glass tube *a* bears a copper scoop at the end, on which is placed a little heap of coal dust. The silica tube *c* passes through the electric furnace *b* and is joined to *a* with a piece of rubber tubing. The silica tube is connected with an apparatus (*f*, *h*, *d*, etc.) for furnishing a puff of gas. After the silica tube is heated by the furnace to 1,100° to 1,200° C., the dust is pushed in to the point *n*, and the blast of gas is suddenly released. This causes ignition of the dust when the system is filled with air.

Pure nitrogen is then substituted for air, and the dust is blown from *n* into *a* in the same way. The atmosphere in *a* is then analyzed for combustible gases.

The results obtained agree with those of the other method. The amounts of combustible gases formed in the inert nitrogen atmosphere under the conditions of test were slight.

THE TESTING OF EXPLOSIVES USED IN MINES.

IMPORTANCE OF TESTING MINE EXPLOSIVES.

The explosives used in coal mines not only cause accidents such as happen in the use of explosives elsewhere, but they have given rise to widespread disasters by igniting explosive mixtures of mine gas and air and of coal dust and air, or both. In addition, the firing of explosives so shakes the walls and roofs of the mines as to cause falls, with their attendant casualties.

In view of the large number of explosives and their wide range in composition and properties, it is obvious that certain of them should prove more suitable for use in mines than others. The problem is to determine which explosives are the most suitable for this purpose and the precise conditions under which they may be used with the greatest safety.

Of course this problem could be worked out in actual mining, but experimental investigations in a working mine are hazardous and slow. To obtain the information sought, in the speediest and most economical manner, laboratory methods are required.

The Bureau of Mines at its Pittsburgh experiment station has equipment for such laboratory testing of explosives. These tests involve chemical and physical examination and the detonation of charges under carefully controlled conditions.

DEVELOPMENT OF PERMISSIBLE EXPLOSIVES.

With the cooperation of explosives manufacturers, the bureau has evolved a type of explosives especially intended for use in gaseous or dusty coal mines. Such explosives, termed permissible explosives,

are tested and approved by the bureau, which also prescribes conditions under which they must be used to be regarded as permissible. Thus the bureau prescribes that an explosive is not to be regarded as permissible unless used with a detonator (blasting cap) of proper strength.

METHODS AND APPARATUS USED IN TESTING EXPLOSIVES FOR PERMISSIBILITY.

The apparatus employed in testing explosives and the methods of using it are briefly outlined in the paragraphs following.

PHYSICAL EXAMINATION.

Every explosive tested for permissibility is examined as to its physical properties, including the determination of the average diameter, length, and weight of the cartridge, whether the cartridge has been dipped more than once in moisture-proofing material, the apparent specific gravity of the cartridge, the weight of the wrapper per 100 grams of explosive, and the color and consistence of the explosive.

When feasible, screening tests are made to determine the sizes of the different ingredients.

Some of the tests made to determine the strength and relative safety of the explosive are described below.

THE BALLISTIC PENDULUM.

The ballistic pendulum, shown in Plate VII, *A*, is used to measure what is termed the deflective force of the explosive. A charge tamped with dry clay is loaded into a small cannon, shown on wheels in the plate. The charge is fired into the bore of a mortar weighing 31,600 pounds, suspended by means of a stirrup from two cast-steel saddles fitting over a steel supporting beam that rests on nickel-steel knife edges. The amount of the explosive being tested that causes the pendulum to swing as far as does a charge of one-half pound (227 grams) of the Pittsburgh experiment station standard 40 per cent "straight" nitroglycerin dynamite is known as the unit deflective charge. It must be less than 1 pound (454 grams). This test indicates the coal-getting strength of an explosive.

GAS AND DUST GALLERY NO. 1.

The liability of an explosive to produce an ignition of coal dust in a mine is determined by firing it in what is known as gas and dust gallery No. 1 (Pl. VII, *B*). This consists of a steel cylinder 100 feet

long and $6\frac{1}{2}$ feet in diameter. In part of the gallery are placed sensitive explosive mixtures of gas and air, coal dust and air, or gas, coal dust, and air. The firing of a charge of the explosive in a small cannon into the gallery is similar to a "blown-out shot" in a mine.

An explosive is considered to have passed the gallery test if no one of the prescribed shots ignites the mixture into which it is fired.

In tests 1 and 3, described below, 10 shots are made. For each shot a charge equivalent to the unit defective charge of the explosive is used; the explosive is placed in its original wrapper, and each shot is tamped with 1 pound of clay stemming.

In test 1, on each trial, a mixture of gas and air containing 8 per cent of gas (methane and ethane) is used.

In test 3, on each trial, 40 pounds of bituminous coal dust is used, 20 pounds being distributed uniformly on a wooden bench placed in front of the cannon, and 20 pounds being placed on side shelves in the second 20 feet of the gallery.

In test 4, five shots, each with a $1\frac{1}{2}$ -pound charge, are fired without stemming. A mixture of gas and air containing 4 per cent gas (methane and ethane) and 20 pounds of bituminous coal dust are used. Eighteen pounds of the coal dust is placed on shelves along the sides of the first 20 feet of the gallery, and 2 pounds is so placed in a part of the gallery inclosed by a paper diaphragm that it will be thrown into suspension by an air current.

RATE-OF-DETONATION APPARATUS.

The rate, or velocity, of detonation of an explosive is determined by firing a charge in a covered pit. The explosive ends of the cartridges are cut off and the cartridges are placed in a sheet-iron tube which is suspended in the pit. Through perforations in the side of the tube two wires pass through the explosive at intervals of 1 meter along the length of the tube. A No. 7 electric detonator inserted into one end of the cartridge file detonates the explosive. Each of the wires passing through the charge is electrically connected to a Mettegang recorder (Pl. VIII, 4) which has a soot-covered bronze drum revolving rapidly at a known speed. When the explosion breaks the electric circuit through each of the wires passing through the long cartridge, an electric spark makes an impression on the drum. By special devices, distances between these impressions can be measured accurately. The final result, based on the distance separating the wire, the speed of the drum, and the distance between the spark points, is expressed in meters per second and is called the rate of detonation. This rate largely determines the shattering effect of an explosive.

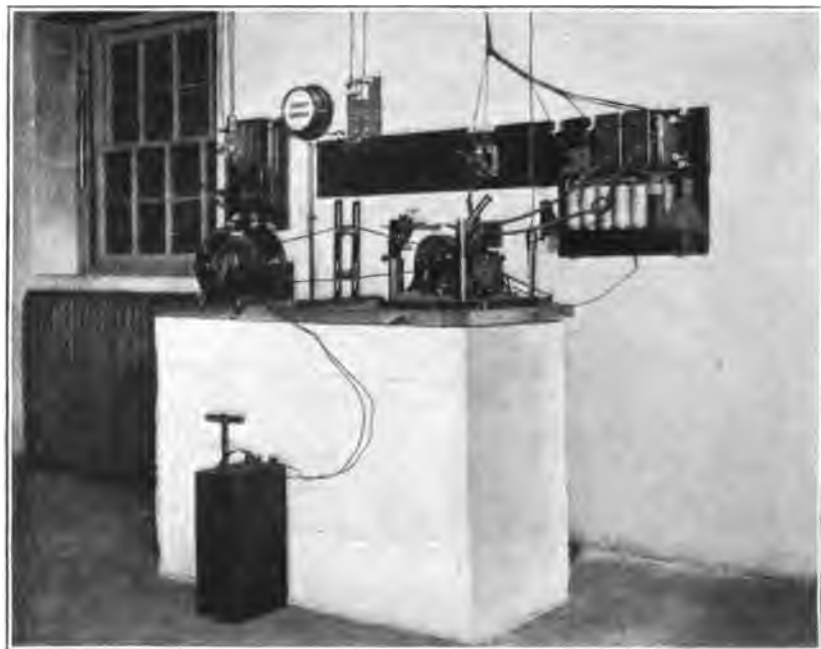
The rate of detonation is determined by three trials with cartridges of the smallest diameter that the manufacturer wishes to use. For



A. BALLISTIC PENDULUM.



B. GAS AND DUST GALLERY NO. 1.



A. METTEGANG RECORDER.



B. BICHEL PRESSURE GAGES AND ACCESSORIES.

a test to be acceptable the entire cartridge file must detonate completely. Similar trials are made with cartridge files having diameters of $1\frac{1}{4}$ inches.

TEST FOR GASEOUS PRODUCTS OF EXPLOSION.

The gaseous products from firing a charge of an explosive are determined by chemical analysis. A charge of explosive is fired in the Bichel pressure gage, described on page 37. The charge is 200 grams of explosive contained in a weight of original wrapper that is proportional to the largest weight per 100 grams of explosive of all the different sizes of cartridge in which the explosive submitted for test is sold. By means of a No. 7 electric detonator inserted into one end, the cartridge is fired after the gage has been closed and the air exhausted from it. The gases evolved by the explosion are allowed to cool for 30 minutes. After the pressure has been recorded, a differential sample of the gases is drawn from the gage and analyzed. If any poisonous gas is present, it is usually carbon monoxide or hydrogen sulphide. The total volume of poisonous gases from 680 grams ($1\frac{1}{2}$ pounds) of the explosive (the permissible charge) is then computed.

An explosive is considered to have passed this test if the total volume of poisonous gases from 680 grams ($1\frac{1}{2}$ pounds) of the explosive does not exceed 158 liters ($5\frac{1}{2}$ cubic feet) at standard temperature and pressure (0° C. and 760 mm.).

If an explosive that did not fulfill the above requirement were used in a narrow coal-mine entry driven ahead of the ventilating current, the air might be so vitiated with poisonous gases as to impair the health of the miner.

PENDULUM FRICTION DEVICE.

The pendulum friction device (Pl. IX, A) is used for testing the sensitiveness of explosives to frictional impact or a glancing blow. Serious accidents have occurred in charging drill holes with explosives too sensitive to such impact. Tests at the Pittsburgh experiment station have shown that some explosives made from mixtures containing chlorate of potash and carbonaceous combustible materials are liable to explode under this test.

These chlorate of potash explosives have many properties making them desirable for use in coal mines; as for example, a slow rate of burning and a heaving effect suitable for the production of lump coal. It would seem, therefore, that there is an opportunity for some manufacturer to produce an explosive of this class that is not sensitive to frictional impact.

An explosive is considered as having passed this test if no explosion, burning, or local cracking occurs in any one of ten trials. In each trial a 7-gram charge of the explosive spread on a grooved

solid steel plate is struck about 18 times by a steel pendulum faced with a fiber shoe (Pl. IX, A). Resting above the shoe is a 20-kilogram weight. The pendulum drops from a height of 1.5 meters.

CRUSHER-BOARD TEST.

A crusher-board test is made of explosives denominated low-freezing, extra low-freezing, or nonfreezing. This test gives the compressive strength of cartridges that have been brought up to room temperature, as compared with the compressive strength of cartridges that have been kept at a low temperature for several hours. Explosives denominated "low-freezing" are kept at 35° F.; those denominated "extra low-freezing," at 0° F.; and those denominated "nonfreezing" are kept at still lower temperatures for several hours prior to the test.

STORAGE TEST.

The two-months' storage test is the final test made in determining permissibility. For an explosive to pass this test at least four out of five cartridges of the smallest diameter submitted must detonate completely when fired after two months' storage at the Pittsburgh experiment station.

In practice the test is made by firing separately five cartridges suspended in air in a gallery. Electric detonators of the grade recommended by the manufacturer are used for the tests. As these tests are considered to be more severe than those made in a cannon, no further tests are made if there is complete detonation in each of five trials.

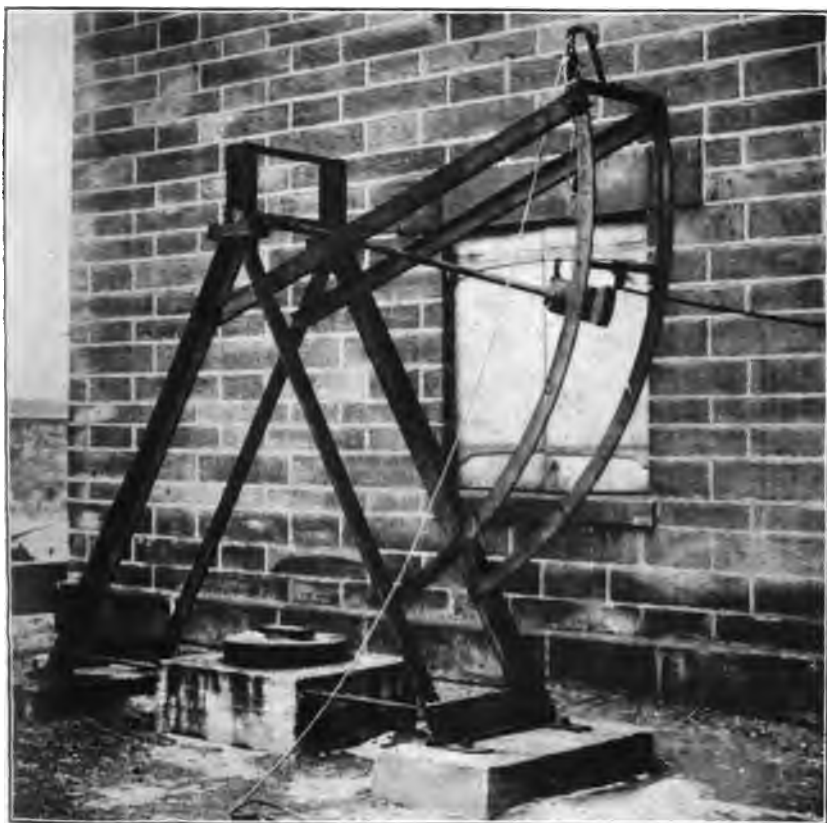
However, if some of the cartridges fail to detonate completely, the number thus failing is noted, and the same number of fresh cartridges are fired separately in a cannon so charged that there is an air space between the bore hole and the explosive; the number and the nature of the failures are noted. Fresh cartridges equal in number to those failing are taken and separately loaded without air space and fired as before. Should there be two or more failures, the explosive fails to pass the test.

SUPPLEMENTARY TESTS.

The explosives that pass all of these tests for permissibility are submitted to further tests in order that the manufacturer and user may have complete information. The additional tests are made as follows:

FLAME TEST.

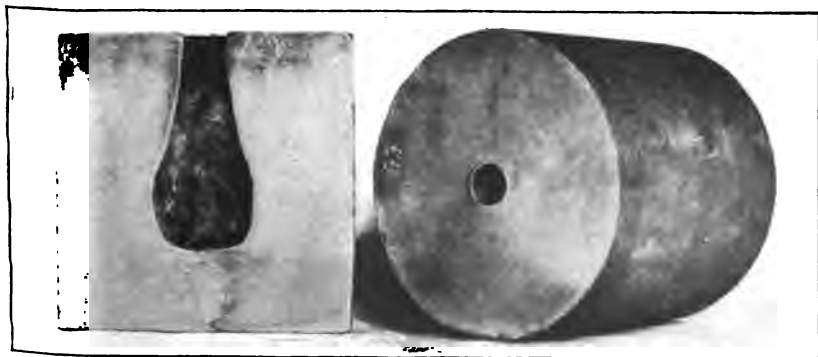
In the flame test the flame from the explosive is photographed in order to show its length and its duration. The test is based upon the belief that the greater the length and the duration of the flame of



A. PENDULUM FRICTION DEVICE.



B. SMALL LEAD BLOCKS. A, BLOCK BEFORE TEST; B, C, D, E, F, BLOCKS AFTER TEST, SHOWING EFFECT OF DIFFERENT EXPLOSIVES.



C. TRAUZL LEAD BLOCK AFTER TEST, AND SECTION SHOWING EXPANSION OF CAVITY BY EXPLOSIVE.

an explosive the greater is the chance of such a flame igniting inflammable or explosive mixtures of gas or coal dust in a mine. Too great emphasis must not be placed on length and duration of flame as an indication of permissibility, for both the length and the duration of the flames of certain nonpermissible explosives are less than those of some of the permissible explosives, but the other variables, particularly the temperature of the flame, are of importance and must be taken into account.

EXPLOSION-BY-INFLUENCE TEST.

The explosion-by-influence test determines the sensibility of an explosive to the explosion wave produced by the detonation of an equal mass of the same explosive and transmitted through the air. In this test two cartridges of the same explosive are placed a predetermined distance apart. One cartridge is fired by means of an electric detonator. If the second cartridge is detonated by the detonation of the first cartridge, the test is repeated with the second cartridge farther away. If the second cartridge is not detonated, the test is repeated with the second cartridge nearer the first. By proceeding in this way, the minimum distance between cartridges at which the first cartridge does not detonate the second is established within 1 inch.

BICHEL PRESSURE GAGE.

The Bichel pressure gage (Pl. VIII, *B*) is used for determining the maximum theoretical pressure that could be developed in a bore hole by an explosive. The essential features of this apparatus-used are cylinders in which the explosive is fired, instruments by which the pressure developed in these cylinders is recorded, and a device by which the heat distribution, or dissipation, is ascertained.

The apparatus actually employed consists of two stout cast-steel cylinders. The chamber of one cylinder measures 19 inches (48.3 cm.) by 7.87 inches (20 cm.); the chamber of the other measures 25.25 inches (64 cm.) by 7.87 inches (20 cm.). The heads of the cylinders are provided with lead gaskets, and are secured in place by heavy stud bolts and an iron yoke, as shown in Plate VIII, *B*. A system of sheaves and suspended counterweights is provided to aid in detaching the heavy heads from the cylinders and in mounting them on the specially designed wagons so that access may readily be had to the interiors of the cylinders.

As the air is practically excluded from a bore hole in coal or rock when it is filled with a charge of explosive, these conditions must necessarily be simulated in experimental tests. Therefore, when explosives are fired in the cylinders the air is almost entirely removed from them with a pump.

The charge of explosive is placed in the gage and fired electrically, and the pressure is recorded by suitable instruments.

BOMB CALORIMETER.

The heat liberated by the detonation of a known weight of explosive is determined in a bomb calorimeter, shown in Plate X, A.

SMALL LEAD BLOCKS.

The small lead block test determines the relative quickness of an explosive, as indicated by the compressive effect exerted by an explosive when exploded unconfined on the upper end of a small cylinder of lead. In making this test, a small steel disk placed on the lead cylinder receives the immediate impact of the detonation. The deformation of the cylinder is taken as a measure of the disruptive effect of the explosive. (See Pl. IX, B.)

TRAUZL LEAD BLOCKS.

The Trauzl lead block test is designed to measure the comparative disruptive effect of explosives fired under moderate confinement. Equal weights of different explosives are confined by means of a fixed quantity of stemming in bore holes of definite size made in lead blocks (Pl. IX, C) of prescribed character and, when thus confined, are exploded by means of similar detonators. The measure of the test is the increase in the volume of the cavity in the block caused by the pressure exerted by the explosive.

IMPACT MACHINES.

The large and small impact machines (Pl. X, B) are used in the determination of the relative sensitiveness of explosives to the impact produced by falling weights.

SUITABILITY TESTS OF EXPLOSIVES.

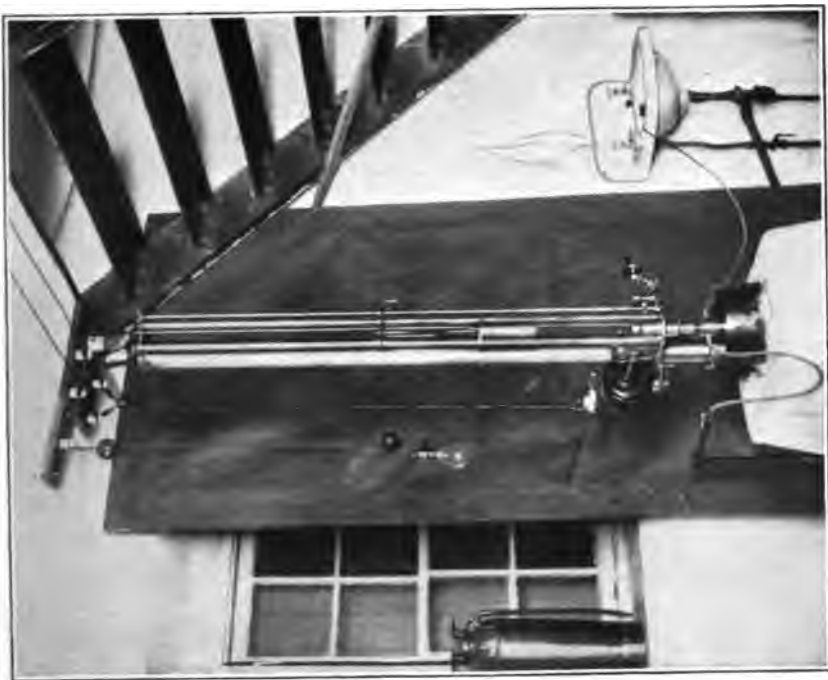
Suitability tests are made to determine the suitability of explosives for use in metal mines and quarries. They are the same as those for permissibility, with the following exceptions:

The ballistic pendulum test is made in duplicate, but no limits are set on the result representing the unit defective charge.

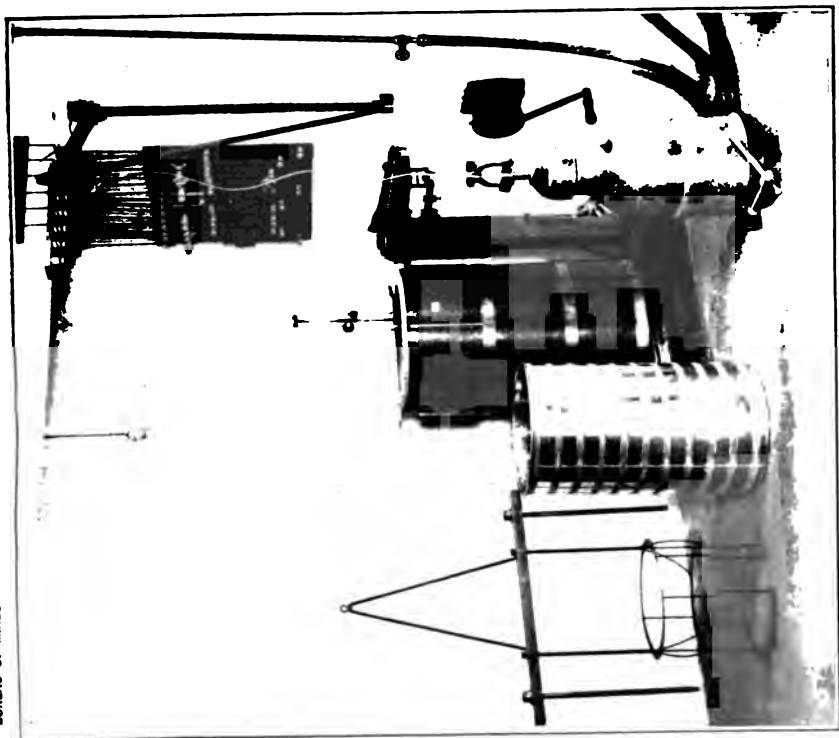
No gallery tests are made.

The rate of detonation test is made in duplicate. The electric detonator used must have a strength not less than that of a No. 6 electric detonator.

In the test for gaseous products of explosion the explosive is fired with an electric detonator of not less strength than that of a No. 6. When detonated in the gage the explosive, to be satisfactory, must not



B. IMPACT MACHINE.



4. CALORIMETER AND ACCESSORIES.

evolve poisonous gases in quantities that may be considered harmful to the health of the miners.

No flame photograph is made.

TESTS OF DETONATORS.

The methods of testing detonators have been described in Bulletin 59.^a The nail test, one of those described, can be readily performed by anybody desiring to determine the relative strength of detonators. This test depends on the bending of a nail by firing a detonator or electric detonator in close proximity to it.

Recently the Bureau of Mines has developed a method of determining the probability of detonation when a detonator is fired. An insensitive dynamite is obtained by mixing 40 per cent "straight" nitroglycerine dynamite with varying quantities of wood pulp. The different mixtures are formed into $1\frac{1}{2}$ by 8 inch cartridges. The detonator to be tested is inserted into the end of one of these cartridges, the cartridge is suspended in the air, and the detonator is fired.

Usually if three cartridges detonate completely, another mixture is used containing less nitroglycerin and more wood pulp. This procedure is continued until such a mixture is obtained that only part of the cartridges prepared therefrom detonate completely. Enough of the mixtures containing the same quantity of nitroglycerin is prepared so that at least 25 trials may be made. The number of cartridges detonating completely out of the total number of trials made with the same insensitive mixture determines the probability of detonation. The results are usually compared with the results obtained with some standard detonator tested at the same time.

SAND TEST.

Another direct test for determining the strength of detonators has been designed by the engineers of the bureau, and is known as the sand test. The detonator is embedded in the center of standard 30-mesh quartz sand contained in a block of steel having a cylindrical cavity. Two cylindrical steel covers are provided—one with a single hole in the center for the passage of a fuse when detonators are tested, and one with two holes for electric wires when testing electric detonators. The device is shown in Plate XI.

The detonator is fired with a fuse or electrically, and the quantity of sand pulverized is taken as a measure of the strength of the detonator. This test, besides being quantitative, enables an experimenter working on reinforced detonators (those in which part of

^a Hall, Clarence, and Howell, S. P., Investigation of detonators and electric detonators: Bull. 59, Bureau of Mines, 1913, 73 pp., 7 pls., 5 figs.

the mercury fulminate or other initiating explosive is replaced by an organic nitro compound or nitrate) to determine whether complete or partial detonation has been brought about, the result being shown by the amount of sand crushed or by the unexploded nitrate or nitro compound found after the shot was fired. The test offers great possibilities for studying the sensitiveness to detonation of explosives and is a distinct aid to manufacturers and users of detonators.

ADOPTION OF PERMISSIBLE EXPLOSIVES AND SAFETY DEVICES.

In its educational campaign for greater safety in mining, the Bureau of Mines has earnestly advocated the use of approved explosives and equipment. Too much stress can not be placed on the need of safeguarding the 1,000,000 men employed in the mining industry in the United States.

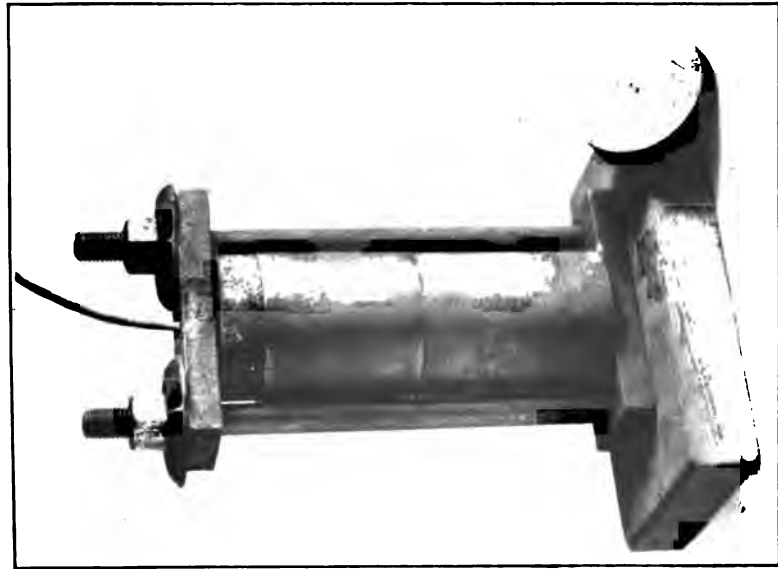
As an example of the importance of adopting safety measures may be cited the decreased fatality rate from explosives in bituminous coal mines in the 7-year period 1909-1915 in comparison with the increased use of permissible explosives. Thus, in 1909, there were 122 fatalities due to explosives, whereas in 1915 the number had been reduced to 76, or almost 50 per cent. In 1909 the amount of permissible explosives used in coal mines was, in round figures, 9,000,000 pounds, and in 1915 the amount used had reached 22,000,000 pounds.

PROGRESS IN REPLACEMENT OF BLACK BLASTING POWDER BY PERMISSIBLE EXPLOSIVES IN ILLINOIS AND INDIANA.

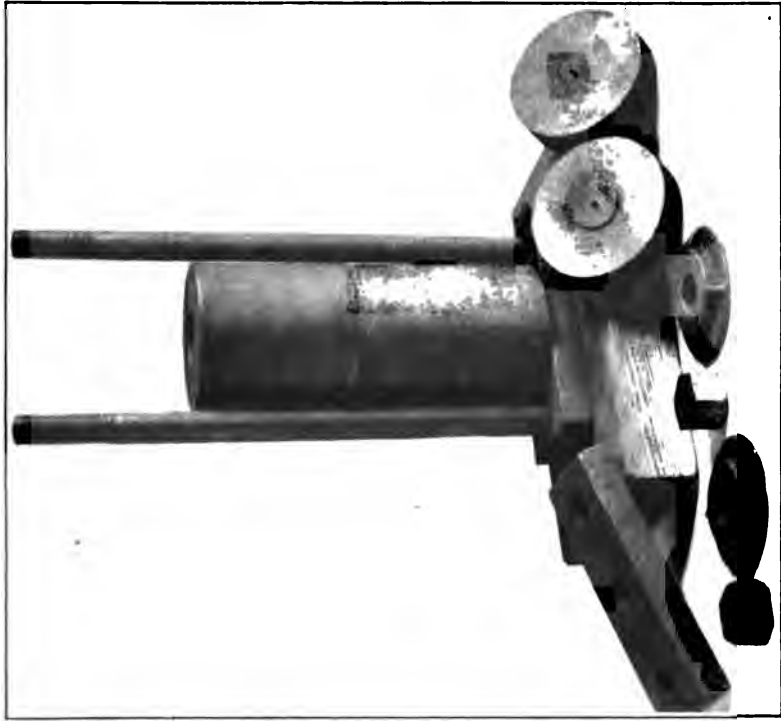
In the coal-mining region designated by the Bureau of Mines as the central district, the bureau has been making special efforts to induce the operators to adopt the use of permissible explosives. It has assigned an assistant mining engineer to the work of education and demonstration. The results in Illinois and Indiana are mentioned specifically as typical.

PERMISSIBLE EXPLOSIVES IN ILLINOIS.

Permissible explosives were not introduced into Illinois until 1909. They are now being used exclusively in 20 mines, which are among the most modern in the State. These mines have a daily output of over 50,000 tons, and employ about 9,000 operatives, or approximately 11.5 per cent of the total coal-mine employees of the State. During the fiscal year ended June 30, 1915, the number of pounds of permissible explosives reported in the Illinois coal reports as used in Illinois coal mines was 1,342,334. Approximately 7,680,000 tons of coal was produced by the use of such explosives, or 14.3 per cent of the State total produced by all explosives.



4. BOMB ASSEMBLED FOR MAKING SAND TEST.



B. DETAILS OF BOMB.

In most of these mines black blasting powder was formerly used, but a change to permissible explosives was made to procure relief from numerous fires and explosions. Thus, in one mine 5,372 fires were recorded in a 14-month period. Since the introduction of permissible explosives, fires and explosions have been practically eliminated and many other advantages have been gained, such as less injurious effects upon the roof, elimination of "blown-out" and "windy" shots, lower cost of explosive per ton of coal, and fewer accidents in handling explosives. At the same time the sizes of coal obtained have not been materially affected. The consensus of opinion among mining men in the State is that where the coal is properly undercut and "snubbed" permissible explosives can be successfully used with little if any increase in the amount of slack coal over that produced by black blasting powder.

The State of Illinois cooperated with the Bureau of Mines in the work, and a report on the use of permissible explosives in Illinois coal mines is to be published by the bureau.

PERMISSIBLE EXPLOSIVES IN INDIANA.

In Indiana, permissible explosives are used exclusively in two mines, one of which is in Sullivan County and the other in Knox County. These mines represent a total daily capacity of 5,000 tons and are thoroughly modern. Permissible explosives were introduced when the screen-coal basis of payment was in force. The results obtained have been entirely satisfactory.

USE OF PERMISSIBLE ELECTRIC LAMPS.

The increased use of electric lamps by miners has been a large factor in increasing safety in mines, particularly in decreasing the possibility of gas ignitions by open lights. Prior to the fiscal year 1916 three types of electric lamps had been given the bureau's approval, having passed the requirements for safety and practicability. Five more approvals were given during the fiscal year, two of which, however, were for lamps of almost identical design. Seven of these approvals are for cap lamps and one for a hand lamp.

The increased use of such lamps has been due in most part to the desire of companies to decrease the hazard of fire-damp ignition by open lights. Prior to the development of miners' electric safety lamps there was usually great opposition to replacement of open lights by oil or gasoline safety lamps because of the great reduction in illuminating power; but the miners' electric safety lamp gives so much more light than the gasoline or oil-burning safety lamp that the old objection to replacement of open lights has been largely overcome. The scale agreement between miners and coal operators in

one large coal-mining district calls for installation of miners' electric lamps in gaseous mines as rapidly as the lamps can be obtained. In one State the law requires the replacement of open lights by permissible electric lamps.

Another factor influencing the installation of miners' electric lamps is the attitude of the insurance companies in decreasing premiums at mines on account of less liability of mine explosions where miners' electric lamps are in use. At the end of the fiscal year 1916, electric lamps were being installed at the rate of 2,500 a week.

IMPROVEMENTS IN MINE HAULAGE.

DANGERS OF ELECTRIC HAULAGE IN DUSTY MINES.

The hazard of electric haulage in dusty mines has not been generally recognized. Wherever there is a possibility of an explosive dust cloud being formed, the presence of electric equipment that may furnish a spark or arc to ignite the cloud is a grave menace to safety. At least one large explosion was caused by the ignition of a dust cloud when a trip was wrecked and an arc resulted from the displacement of the trolley wire and the consequent short circuiting of the current.

Great care should therefore be taken to see that the coal dust on haulage roads has been made inert either by wetting or rock dusting, as described on previous pages.

Care should also be taken to see that the trolley wire is securely supported at short enough intervals so that displacement of wire is less likely in the event of a wreck.

CHARACTER OF IMPROVEMENTS.

Probably the greatest change in mine haulage equipment in recent years will result from the development of the storage-battery locomotive. The claimed advantages for its use are the increased safety resulting when it replaces electric haulage using trolley wires, and the lower cost and increased speed of haulage resulting when it replaces mule haulage. With respect to safety, the elimination of the trolley is unquestionably an increase in safety. However, the storage-battery locomotive in its present development should not be regarded as being safe for use in gaseous mines. The possibilities of sparking or arcking are numerous, and therefore ignitions of gas accumulations, should such accumulations occur, are quite possible. In addition there is an added menace due to the evolution of hydrogen in the batteries. It is known that at least one local explosion in a charging station was due to the ignition of gas evolved from batteries.

The Bureau of Mines is now working on specifications for making storage-battery locomotives explosion proof, and it is hoped that the development of locomotives that will be practically safe for operation in gaseous places will result.

Many companies at the present time are making comparative tests of the efficiency and cost of storage-battery locomotives and other forms of haulage. Should such tests generally result favorably, a great advance in the importance of the storage-battery locomotive industry will undoubtedly result.

COOPERATIVE WORK IN CALIFORNIA.

Mine inspection in the State of California is carried out under a cooperative agreement between the Bureau of Mines and the industrial accident commission of the State, whereby a mining engineer of the Bureau of Mines occupies the position of chief mine inspector of the State. He is assisted by an able corps of deputies who are appointed by the State after an examination to establish their qualifications. California is the only State in which mine inspection is carried out under such an arrangement. The work of the mine inspector essentially concerns metal mining, because practically no coal is mined in California. There are in all approximately 14,000 men employed in mines, quarries, and ore-treatment plants in the State.

In 1915 the bureau's engineer drew up a set of tentative mine safety rules, copies of which were printed and distributed throughout the State. Operators, miners, and those interested in mining were given a chance to comment on these rules at public hearings, after which a final draft was made and adopted by the industrial accident commission, to become effective January 1, 1916.

The workmen's compensation, insurance, and safety act of the State of California gives the industrial accident commission the power to make and enforce safety rules and regulations, to prescribe safety devices, to fix standards, and to order the reporting of injuries. Thus, the mine safety rules, although not enacted into law by the State legislature, are in effect the same as laws, for an offender against these rules renders himself liable to prosecution for maintaining an unsafe working place.

The mine safety rules, as adopted by the commission, cover every detail of operation in and about mines. They may be said to be somewhat drastic, but are tempered by a provision that "in cases where, in the opinion of the industrial accident commission, the enforcement of any rule would not materially increase the safety of employees, and would work undue hardship on the operator, exemptions may be made at the discretion of said commission, but such

exemptions must be in writing to be effective, and can be revoked after reasonable notice is given in writing; provided, further, that the rules shall not apply to the operation of mines employing three men or less on one shift, or to gold dredges, hydraulic mining operations, or surface placer mining, except where the rules specifically provide for the inclusion of these classes of mining in their provisions."

At the beginning of 1916, when the mine safety rules went into effect, the safety movement was young in the State of California. Mine operators were unused to State supervision and in some respects many of the mines were not being operated in accordance with these rules. It would have been impossible immediately to change conditions in the mines. The commission adopted the policy of meeting the operators halfway, being reasonable in its demands, and affording all assistance practicable for the betterment of conditions.

It was thought possible that there might be objection to many of the rules and that some dissatisfaction might result. However, the months during which the rules have been in force have brought forth surprisingly few complaints. Furthermore, it has been necessary to grant only few exemptions under the rules. The work of improving conditions and of installing safety devices, as provided by the rules, has gone forward steadily and rapidly, and the operators of the State are to be congratulated on the cooperation that they have given the industrial accident commission.

During the first six months there was not a single case where it was necessary to invoke the law because of the refusal of any operator to abide by the mine safety rules. Operators generally agree that there has been a vast improvement in conditions.

The chief mine inspector, in an effort to be of assistance to mine operators, issues a monthly bulletin pertaining to safety and efficiency in mines. In this bulletin is published a list of sketches of safety and efficiency devices that may be obtained by California operators free of cost. In addition, a campaign has been started among the miners to procure their cooperation in preventing mine accidents. The efforts in this direction embrace an organization known as "the miners' safety bear club." This club was organized for the purpose of getting in close personal touch with the miner so that information of value could be imparted to him. This club was organized in March, 1916, and by the end of June it had nearly 5,000 members. Monthly letters containing valuable information regarding accident prevention are written to the miners.

From January 1 to June 30, 1916, under the operation of the mine safety rules the number of deaths, as compared with those during the first half of 1915, materially decreased.

It is probable that in the course of a year a few minor changes may be made in the rules. As a whole, the system of mine inspection appears to be sound, and without question it has resulted in the prevention of many serious and fatal injuries to the miners of California.

STATISTICS OF ACCIDENTS IN THE MINERAL INDUSTRIES.

To enable the Bureau of Mines efficiently to carry out its duty of increasing safety in the mineral industries it was early evident that the causes of accidents would have to be studied in order that effective remedies and safeguard could be devised. Accordingly the bureau began the compilation of statistics of accidents in the mineral industries.

When the bureau was established no accurate and comparable statistics of mine accidents were available for all the mining States. Many of the States had an inspection service, but as each State had a different system of classifying accidents the collected figures were not precisely comparable and could not be readily used as a basis for efforts to increase safety in mining.

Limited funds have restricted the bureau's efforts to collect accident statistics, but through the hearty cooperation of State mine inspectors it has been able to compile an annual statement of coal-mine fatalities and, through the cooperation of mine and quarry owners, to publish statistics of metal-mine and quarry accidents. The bureau has also compiled annual statements of the production and distribution of explosives by States and has collected accident statistics for coke ovens, ore-dressing plants, and smelters.

Under a cooperative arrangement with all State coal-mine inspectors a monthly report of coal-mine fatalities, showing the number, cause, and distribution by States, is published shortly after the close of each month.

All of the coal-mine fatalities (over 50,000) reported by State mine inspectors since the beginning of inspection by each State have been tabulated by causes, calendar years, and States, the figures covering the mining of more than 89 per cent of all the coal produced in the United States since 1807. A bulletin containing these figures placed for the first time all reported coal-mine fatalities on a calendar-year basis under a uniform classification.

The bureau has records covering a five-year period for metal mines and quarries and a three-year period for coke ovens and metallurgical plants. Although based on reports voluntarily furnished by the operators, the reports present reasonably accurate data from which to devise plans whereby accidents may be reduced and the mines made safer. The figures, although not absolutely complete, are directly

comparable, being on the same basis and by calendar years for each State. This uniformity is highly desirable, as it permits intelligent comparison of the figures for the various States.

During the fiscal year 1916 a bulletin, mentioned above, on coal-mine fatalities in the United States from 1870 to 1914, was issued. In this bulletin are classified 53,000 fatalities as obtained from a systematic search through the various State mine inspectors' reports. Although it is possible that the records during the earlier inspection periods were not complete, yet sufficient data have been accumulated to determine the principal causes of coal-mine accidents. Accidents as related to coal mining by machines have also been studied and tabulated in detail in Bulletin 115.

Data are still lacking concerning the number of nonfatal injuries in the coal mines, but sufficient material has been collected in the metal-mining industry to allow a fair estimate of what may be expected in the coal mines. In order that more detailed information concerning the nonfatal injuries may be obtained, it will be necessary to develop further the cooperative agreement with the State mine inspectors and State compensation bureaus so that such information may be obtained in addition to data regarding the coal-mine fatalities now reported by the State mine inspectors.

Some States do not require a record of injuries or a report to any individual, insurance board, commissioner, or inspector, whereas other States have strictly enforced laws requiring such reports. Furthermore, the fiscal years are not uniform. The bureau in its efforts to procure more of the data above mentioned called a conference of State mine inspectors, Federal officials, and representatives of compensation commissions and insurance companies in Washington on February 24-25, 1916. A tentative schedule was prepared for uniform mine statistics, in which, among other things, reporting of accidents by calendar years was recommended. This schedule and a report of the conference have been distributed to the inspectors, mining companies, and others interested in mining statistics, with a request for further suggestions and recommendations. These recommendations will be taken up by a special committee, and it is hoped that this committee will be able to recommend some standard forms for mine statistics which may ultimately be adopted by each and every State interested in mining.

In addition to the monthly reports on coal-mine fatalities, a report on accidents in metal mines^a and a report on accidents in quarries,^b each for the calendar year 1914, were published; also a report on the

^a Fay, A. H., *Metal-mine accidents in the United States during the calendar year 1914*: Tech. Paper 129, Bureau of Mines, 1916, 96 pp., 1 pl., 3 figs.

^b Fay, A. H., *Quarry accidents in the United States during the calendar year 1914*: Tech. Paper 128, Bureau of Mines, 1915, 45 pp.

production of explosives during 1915^a and a report on coke-oven accidents in 1915^b were issued.

SUPERVISION OF MINES ON SEGREGATED INDIAN LANDS.

BUREAU OF MINES CHARGED WITH INSPECTION.

Under date of February 11, 1913, the Secretary of the Interior issued an order charging the Bureau of Mines with the inspection of the physical operation of mines on Indian lands in the United States, with special reference to mines in lands belonging to the Choctaw and Chickasaw Indians of Oklahoma and known as segregated Indian lands. Accordingly the bureau established an office at McAlester, Okla., and placed it in charge of an experienced engineer with instructions to study the various conditions existing in that field and to draw up suitable rules and regulations covering the operation of the mines on the Indian lands, and also to cooperate with the Office of Indian Affairs in passing upon applications for leases on the Indian lands, upon applications by the mining companies to reserve from sale certain parts of the surface of their leases for use in mining operations, and upon other questions relating to the rights of the Indian lessors.

Most of the applications for reservations of surface were filed and passed upon prior to the beginning of the fiscal year 1916, but just prior to the offering for sale of the remainder of the surface of the segregated lands in December, 1915, some of the lessees made application either for additional reservations or for changing the existing reservations, relinquishing part and taking other lands instead. These reservations were made as requested. During the year operating coal companies made several applications to purchase the surface of the segregated coal lands.

PERMISSIBLE EXPLOSIVES ORDER.

On May 4, 1914, the Secretary of the Interior issued a further order requiring that "on and after August 1, 1914, only such explosives as * * * have been designated 'permissible explosives,' shall be used in any coal or asphalt mine on the segregated coal and asphalt lands * * * in Oklahoma." The order further provided that explosives other than permissible explosives might be used if fired by an electric system from without the mine.

^a Fay, A. H., Production of explosives in the United States during the calendar year 1915, with notes on coal-mine accidents due to explosives and list of permissible explosives, lamps, and motors tested prior to May 1, 1916: Tech. Paper 159, Bureau of Mines, 1916, 24 pp.

^b Fay, A. H., Coke-oven accidents in the United States during the calendar year 1915: Tech. Paper 161, Bureau of Mines, 1916, 18 pp.

The time for taking effect of this order was afterwards extended to January 1, 1915.

In accordance with an agreement reached at a conference held in Washington in April, 1916, between the Director of the Bureau of Mines and a committee of Oklahoma coal operators, a number of requests for temporary exemption of mines from the operation of the permissible explosives order of the Secretary of the Interior have been received; and as rapidly as possible the mines mentioned will be reexamined and reports rendered.

On the whole, the permissible explosives order has produced very good results, although these results are necessarily slow in developing. There have been fewer shot-firer accidents and fewer mine fires from explosives used than in previous years.

SUPERVISION OF MINES ON ALLOTTED INDIAN LANDS.

During the fiscal year 1916 there were 42 formal and informal conferences held with officials of the Indian Office in regard to proposed and present operating mines on lands allotted to individual Indians as opposed to segregated Indian lands allotted to the two tribes previously mentioned. In about 90 per cent of the instances it was necessary to make field inspections in order to ascertain whether the bonus and royalty provided were sufficient. Most of these inspections are made jointly with the chief oil and gas inspector of the Five Civilized Tribes, who submits a report to the Indian Office after consulting the representative of the Bureau of Mines.

The leases on allotted Indian lands cover the mining of a wide variety of metallic and nonmetallic minerals, including lead and zinc, gold and silver, glass sand, gravel and sand, coal, and asphalt. The leases for precious metals have not as yet resulted in the development of anything but prospect holes. The same is true of the lead and zinc leases outside of the Miami lead and zinc district in north-eastern Oklahoma. The coal and asphalt leases cover coal leases exclusively, as practically all of the important asphalt deposits are on segregated lands. The coal mines on allotted Indian lands are being inspected as fast as possible.

In all, applications for examination of 22 tracts of allotted lands were received at this office during the fiscal year 1916. These applications were divided as follows:

Coal and asphalt, 13 tracts, embracing a total of 1,500 acres; application for removal of restrictions, 4 tracts, embracing a total of 200 acres; application for sand and gravel lease, 1 tract, embracing 280 acres; gold, silver, copper, lead, and zinc leases, 2 tracts, embracing a total of 60 acres; application for report regarding damage

to surface by water from mine workings, 2 tracts, covering 340 acres. In all a total of 2,380 acres were examined. In addition, an examination was made at the request of the probate attorney at McAlester, Okla., as to the advisability of canceling two leases held on allotted lands.

Although these tracts of land are small, and do not usually require much time for an inspection, yet they are situated in widely scattered parts of the State, and an inspection trip to a given tract and return to McAlester, Okla., may require the greater part of a week.

WATER, OIL, AND GAS WELLS ON THE SEGREGATED COAL LANDS.

Although the regulations under which the surface of the segregated coal lands was sold provided that the locations for all water wells, as well as for oil and gas wells, be approved by the representative of the Bureau of Mines and the State mine inspector, few applications have been filed asking for such approval. There have been to date only three applications for approval of water wells, all of which have been passed upon. These wells are only 20 to 30 feet deep and are unimportant.

Under date of May 13, 1915, application was made to the Bureau of Mines for permission to drill an oil and gas well on the segregated coal lands near Coalgate, Okla. There were no mines, however, within several miles of the place where drilling was contemplated, and a diamond-drill hole put down some years ago showed no coal within a half mile of the proposed well. This information was brought out as a result of a careful inspection. The application was approved, the approval being conditioned upon the applicant submitting to the district engineer of the bureau a log of the well, showing the different strata through which the well passed, in order that it might be determined whether coal existed under this part of the segregated lands. The drilling resulted in a dry hole. There are at present no oil or gas wells on the segregated lands, though on the allotted Indian lands in the Henryetta district there are oil and gas wells. The protection of working mines from gas wells is given careful attention by the bureau's McAlester office, in cooperation with the State inspector of mines and the inspectors of the petroleum division of the bureau. A written form of agreement covering oil and gas well drilling operations on allotted Indian coal lands, to be signed by both the coal-land lessees and the oil and gas lessees, has been prepared, in cooperation with attorneys, and is usually readily entered into by both coal and oil and gas lessees when their leases cover the same tract of allotted Indian lands.

SUPERVISION OF OIL AND GAS LEASES IN RIVER BEDS.

During the fiscal year 1916 the district engineer of the bureau for the southwestern coal-mining district has continued work as representative of the Federal Government on the supervisory committees appointed by the Federal judges in the eastern and western judicial districts of Oklahoma. This committee has charge of the physical operation of oil and gas wells operating in the beds of the Cimmaron and Arkansas Rivers, pending the suits to determine the rightful owners of the oil. The State of Oklahoma and the Federal Government each has one representative on the supervisory committee. In each judicial district there is a receiver appointed to receive the royalty in dispute.

This work has produced large returns in cash, although the time required of the district engineer to do this work has at no time amounted to more than two to four days a month.

MINE INSPECTION IN ALASKA.**GENERAL CONDITIONS IN THE TERRITORY.**

Shortly after the Bureau of Mines was established in 1910, the Secretary of the Interior authorized the bureau to take charge of the inspection of mines in Alaska, and Congress has for several years provided for the employment of a Federal inspector of mines for Alaska who is under the supervision of the Director of the Bureau of Mines and renders an annual report to the Secretary of the Interior through the director.

The report for the fiscal year 1916 shows that the year was particularly prosperous for the Territory, owing to the war demand for its metals and fish, the construction of the Government railroad, the opening of the coal lands to lease, the taking up of considerable areas by homesteaders, and a particularly long and dry summer which favored crop growing during 1915.

The war demand which raised the price of metals forced the production of the operating mines far beyond the ordinary figures, caused the reopening of others that had been idle since the fall of the price of copper in 1907, and stimulated both capitalists and prospectors to discover promising prospects.

The construction work on the Government railroad gave employment to several thousand men and assured transportation to remote parts of the Territory, previously almost inaccessible on account of the excessive cost of freighting. The railroad is thus opening new fields both to the miner and to the farmer.

The opening of coal lands outside the Bering River and Matanuska coal fields for free-use 10-acre permits has offered a cheap coal for

local use, and, although the larger tracts in the Bering River and Matanuska fields have only recently been laid out in units for leasing, it is hoped that the development and production of coal in these fields will be rapid, leading to the manufacture of coke and eventually to the erection of smelting and other plants.

FIRST-AID AND MINE-RESCUE ORGANIZATION.

First-aid training made rapid progress at Juneau, where several hundred miners were given more or less instruction. At the Panama-Pacific Exposition, during September, 1915, a team representing a large gold-mining company tied for fourth place in competition with 25 other teams, many of which had had previous experience in competitive work and years of training.

MINERAL PRODUCTION.

The production of placer gold fell off slightly during the fiscal year, owing to the exhaustion of some of the richer placers, but the lode gold output was considerably increased by the opening of large low-grade properties near Juneau. The production of copper more than doubled in tonnage and trebled in value. Antimony, which is found associated with the lode gold of the Fairbanks district and previously was shipped only intermittently, was eagerly sought by ore buyers, and tin, marble, and gypsum ran about the same as the year before. A small amount of oil was produced at Katalla, but lack of transportation facilities and financial difficulties prevented the company operating there from producing on a scale of any magnitude.

The mineral output of Alaska in the calendar year 1915, according to the United States Geological Survey, includes the following totals: Copper valued at \$15,139,129; gold valued at \$16,702,144; silver valued at \$543,398; tin, lead, antimony, marble, gypsum, coal, and petroleum valued at \$469,503.*

COAL LEASES.

Local users of coal in the immediate vicinity of the various coal fields have been prompt to take advantage of the act of October 20, 1914, authorizing the issuance of permits to mine coal, without payment of royalty, from 10-acre tracts. Some two dozen permits have been granted. These cover areas from Juneau to Candle Creek, north of Nome, with the majority along the shores of Cook Inlet. Operations have also been resumed on groups of patented ground in the

* Preliminary estimates by the Survey show the following figures for the calendar year 1916: Copper, \$32,400,000; gold, \$17,050,000; silver, \$950,000; lead, tin, antimony, tungsten, petroleum, marble, gypsum, and coal, \$350,000.

Bering River field, and private capital is to construct a railroad to that field.

To furnish a supply of fuel for the Alaskan Engineering Commission, a 5-foot bed on the west bank of Moose Creek in the Matanuska field, about a mile and a half upstream from the railroad right of way, was opened.

The Bering River and Matanuska fields were opened for leasing in May, 1915, and the time for the return of bids was extended into the fiscal year 1916.

GOVERNMENT RAILROAD.

The Government railroad under construction from the coast to the interior has been laid to the central part of the Matanuska coal field and will be advanced along the main line or by spurs. The docks at Seward were rebuilt and the Alaska Northern has been rehabilitated between Seward and Kern Creek. Track has been laid about 15 miles southeast of Anchorage to Potter Creek on Turnagain Arm. From Matanuska Junction 30 or 40 miles of track has been laid along the main line, and in the Susitna Valley between Montana Creek and Indian River over 70 per cent of the right of way has been cleared. Work at Nenana has proceeded up the Nenana Valley to unite with the line coming up the Susitna at Broad Pass and up the Tanana Valley toward Fairbanks.

WORK OF THE INSPECTOR.

The mines tributary to Cook Inlet, Prince William Sound, the Copper River, Juneau, and Ketchikan were inspected during the fiscal year 1916. An examination was made of the Matanuska coal field during the summer of 1915 to subdivide the field into leasing blocks, and the Federal inspector made a trip to Washington, D. C., early this spring in connection with the same work. While in Washington the inspector attended the safety-first exhibition as a delegate from the Territory. His reports on the mining and mine inspection in the Territory for the calendar years 1915 and 1916 are in course of publication by the Bureau of Mines.

PROSECUTIONS.

One case for not reporting serious accidents was brought in the name of the Territorial inspector by the Federal and Territorial inspectors. The case charged a mining company with not reporting a gas explosion in which two men were seriously burned. The court dismissed the defendant, as the Territorial law did not define a serious accident.

SAFETY, SANITATION, AND HEALTH INVESTIGATIONS IN METAL MINES.**PULMONARY DISEASE AMONG MINERS IN JOPLIN DISTRICT, MISSOURI.**

During the latter part of 1914 it became possible to take up, in cooperation with the United States Public Health Service, an investigation of siliceous dust and its effects on the health of miners in the Joplin district, Missouri. Complaints that had been made indicated that a large percentage of the miners were suffering from tuberculosis as a result of breathing mine air containing large quantities of dust.

A mining engineer of the bureau and a passed assistant surgeon of the United States Public Health Service made a preliminary survey of conditions during the latter part of 1914. Their activities embraced mines in Jasper, Lawrence, Newton, and Greene Counties, Mo., and outlying districts in Kansas and Oklahoma. It soon became apparent to the investigators that the trouble was chiefly confined to what is known as the sheet-ground mines in Jasper County, so that the greatest amount of time was devoted to this particular district.

While the representative of the Public Health Service devoted his time chiefly to determining the prevalence of pulmonary trouble, the bureau's mining engineer visited the principal sheet-ground mines for the purpose of studying mining conditions and determining the quantity of dust that existed in the mines by reason of the various operations of mining.

The investigators procured the active cooperation of State and county officials and of various societies and individuals. The mining engineer was aided by the three State deputy mine inspectors of the district. At the suggestion of the investigators an organization known as the Southwestern Missouri Mine Safety and Sanitation Association was perfected.

On the completion of the preliminary investigation a report was made setting forth the causes of dust in the mines and offering suggestions for the improvement of conditions. It was also recommended that laws be enacted providing for the use of dust-abating devices, and that an intensive educational campaign be carried on among the miners and their families with a view to pointing out the dangers of breathing the siliceous dust in the mines.

At the request of many prominent operators of the district, the two investigators returned to the district early in February, 1915, for the purpose of assisting in an educational campaign and pursuing the investigation of underground and surface conditions. A clinic was opened and miners were examined and advised free of cost by the representative of the Public Health Service. The bureau's engineer

remained in the district until July 19, 1915, during which time he was enabled to make a thorough investigation of the details connected with the production and abatement of siliceous dust in the mines. He also took an active part in the educational work, addressing, in small groups, practically every miner in the sheet-ground district, calling their attention to the dangers of breathing the dust and soliciting their cooperation in bettering conditions. Public meetings were held, at which a total of 2,700 miners were addressed. At these meetings moving pictures pertaining to sanitation and safety and slides showing magnified particles of siliceous dust were exhibited.

In the end the active cooperation of a large proportion of the miners was obtained. At first they were inclined to regard lightly the warnings and suggestions offered, but when they learned, through the agency of the many addresses made to them, the dangers of the situation, they were not long in getting behind the movement to remove the cause of the trouble. In March, 1915, the Missouri State Legislature enacted into law bills providing for better sanitary conditions in and about the mines, the installation of spraying devices, and the building of sanitary wash and change houses.

The outstanding features and results of the investigation were as follows:

Although miner's consumption was due to the siliceous dust, many other factors were responsible for the consequent tuberculous infection, the chief of these probably being poor living and housing conditions. Seven hundred and sixty-one premises occupied by miners were visited and revealed that overcrowding, lack of cleanliness, and especially an absence of sanitary privies were common.

It was determined that siliceous dust existed in varying quantities in all of the sheet-ground mines, and that this dust was composed of particles of flint that contained usually more than 90 per cent of siliceous residue. The dust particles are sharp edged, often blade-like or knife-like in shape. Owing to their chemical composition they do not dissolve after having been taken into the lungs. This fact, together with the shape of the dust particles, makes it possible for them to penetrate the cell walls of the lung tissue, thus preparing a place for the lodgment of tubercular germs.

The size of the siliceous dust particles was studied. A number of microscopic slides were made from the sputum of miners afflicted with miner's consumption, in which the siliceous dust could be plainly seen. The majority of dust particles were 3 to 5 microns in diameter and smaller, hardly any being larger than 8 microns; it would appear then that it is the very fine dust that is dangerous, because only these fine-sized particles can penetrate into the lung cells, a conclusion in harmony with that reached by investigators in South

African mines, who declare that no particles larger than 22 microns in diameter permanently affect the lung tissue.

A preliminary report was published by the bureau, and the final report will be published at an early date.

The results of this work have been of great benefit to the miners and operators of the Joplin district. Although the work done resulted in intensive efforts to remove the cause of the trouble, it is not to be expected that the disease will be immediately eradicated. Several years may pass before the real benefits will be felt. The immediate results have been a decided improvement in sanitary conditions and almost a complete elimination of the cause of the trouble.

A great factor in the results obtained was the educational work that was carried on, not only among the miners, but also among the operators. The miners were shown the dangers of breathing siliceous dust, and the operators were convinced that poor sanitary conditions and dusty mines resulted only in making their employees unfit for the arduous tasks that miners are called upon to perform.

SILICOSIS AMONG MINERS IN BUTTE, MONTANA.

In May, 1916, the Bureau of Mines, in cooperation with the Public Health Service, undertook, in the mines of Butte, Mont., an investigation into health hazards, particularly silicosis. It was thought that the investigation would closely parallel that carried into effect in the Joplin region, but as the investigation has developed its scope has necessarily widened, and it is probable that before definite conclusions can be reached careful study must be made not only of dust conditions and of methods of allaying the dust, but also of problems of ventilation, heat, humidity, etc., which appear to have a direct bearing on health conditions in general as well as on silicosis. There are approximately 40 large mines in Butte, many of which have men working to depths as great as 3,300 feet, and there are approximately 15,000 miners in the district. Progress reports are being made from time to time, and a final report will be made on completion of the investigation.

HOOKWORM INFECTION IN THE DEEP MINES OF THE MOTHER LODE, CALIFORNIA.

As a result of the work of several practicing physicians, it became evident that some of the miners in the deep gold mines of the Mother Lode in California were infected with hookworm. As hookworm infection is a communicable disease, the State board of health was desirous of stamping it out, and as the infection in mines might be regarded as an industrial injury, the industrial accident commission of the State was anxious to eradicate it. The Bureau of Mines

has been charged with investigations looking to improving conditions that impair the health of miners, and as miners who become infected in one State may spread the infection to miners in other States, the bureau was doubly interested in preventive measures. The three agencies mentioned arranged for a cooperative investigation of hookworm infection among miners in the California mines. The largest share of the expense of the investigation was borne by the California State Board of Health; most of the field work was done by the Bureau of Mines.

SERIOUSNESS OF HOOKWORM INFECTION IN THE DISTRICT.

The investigation showed that the infection, although widespread, is at present, in many cases, not far advanced. The hookworm is an insidious parasite. It does not actually kill, but extends its influence in a quiet, cumulative manner, and in different directions. A man can not enjoy normal health while the worms are hooked to the inside of his intestines and daily sucking his blood. This fact was illustrated by the testimony as to improved physical condition given by some of the miners after taking the hookworm treatment. It should be remembered that although a miner with hookworm may not be seriously affected himself, there is always danger of his spreading the infection if he commits a nuisance in a place favorable to the development of the eggs.

Fecal specimens from 1,096 of the working men from eight Mother Lode mines were microscopically examined for hookworm eggs. Of the 1,096 examined 337, or 31 per cent, were infected with hookworms.

CAUSES OF INFECTION.

A report of the investigation, now in course of publication by the Bureau of Mines, shows that the source of all hookworm infection—discharges from the bowels—is spread by inadequate or inefficiently maintained toilet facilities, and that hookworm larvæ can gain entrance to the body through the mouth or through the skin after it has been broken, and can bore through the intact skin at whatever place infected dirt happens to come in contact with the body. A miner can not do his work without getting dirt on his hands and clothing; therefore his safety lies in keeping the dirt of the mines as noninfectious as possible.

RECOMMENDATIONS FOR PREVENTION.

As a result of the investigation a number of recommendations for preventing future infection of the mines and for minimizing dangers from existing infectious areas were made. These provide for physical examination of all mine employees in the Mother Lode mines; treatment of infected miners; issuance of a "hookworm certificate"

showing a miner to be free from the infection; detailed improvements of toilet facilities; extermination of rats from the mines; sanitary eating places; sanitary drinking fountains or sanitary water kegs; effective mine drainage; improved wash and change houses; and instruction of the miners as to the causes, nature, and prevention of hookworm.

INVESTIGATION OF LOW-GRADE COMPLEX ORES IN THE SILVERTON DISTRICT, COLO.

In response to numerous requests for assistance in evolving suitable processes for treating the low-grade complex sulphide ores of the Silverton district, Colo., the bureau undertook an investigation of these ores. Late in May, 1913, a junior geologist of the Bureau of Mines was sent to the Silverton district to conduct the investigation. The ore deposits are vast and the problems to be solved are complicated by the fact that the ores represent the lean stumps of previously worked bonanza ore bodies.

These are for the most part intimate mixtures of all the common sulphides carrying appreciable values of silver and at times gold, and hence require both concentration and separation. On account of their complex nature, mill recoveries and separations have been at many mines so poor and imperfect that the mines could not be profitably operated. The purpose of this investigation is to get at the relations of the minerals in the ores and the manner of occurrence of the precious metal content in order that the causes of the difficulties encountered in milling may be ascertained and means found whereby the recoveries can be increased and the separations improved. If this can be accomplished, it will make possible the reopening and profitable working of many mines in that region that are now idle. Moreover, the problem is not restricted to the region in which the investigation is being made, but is one that is encountered in many of the western mining districts, and its solution will consequently yield as widespread benefits.

The bureau's representative has carefully studied the conditions in the district and has collected many samples of the various ores. From these samples polished sections were made for a metallographic study, which has already yielded much valuable information as to the composition and character of the ores and the reasons for the low recovery obtained by the concentrating methods used. The next step will be investigation to ascertain the proper means of concentration. Though the bureau is somewhat handicapped by a lack of facilities for such work, the investigations are expected to yield suggestions of practical value to the operators of the Silverton district and to owners of similar ores in all parts of the United States.

COPPER MINING IN NEW MEXICO.

It is a far cry from the unsystematic and haphazard methods of early mining to the carefully organized, strongly financed, and efficiently planned mining efforts of to-day. Many large mining companies now work as systematically as the best manufacturing enterprises. Study and the publication of results obtained inevitably help the mining industry as a whole. Problems solved in one mining district mean that some other district will become prosperous.

In Bulletin 107, entitled "Prospecting and mining of copper ore in Santa Rita, N. Mex.," the bureau presents the results of a study of a most important phase of metal mining, namely, the open-cut method, where steam shovels are employed to load the ore.

CODIFICATION AND ANNOTATION OF MINING LAWS.

One of the activities of the bureau has been the collection, codification, and annotation of the various American and foreign mining laws, statutes, and decisions.

It was believed that the bureau could well serve the mining industry by a complete collection and a systematic grouping of the mining statutes. The United States statutes relating to the location of mining claims on the public domain have been enforced since 1872. Various amendments have been added from time to time, and all these have been variously construed by the several courts, Federal and State. Up to the time of the organization of the bureau no effort had been made to collect and arrange these various acts except in the United States Revised Statutes of 1878, and practically nothing had been accomplished in the way of a general codification of the original statutes and the various amendatory acts with the construction placed thereon by the various courts. The chaotic condition of these mining statutes was regarded as a handicap to the metal-mining industry, as the location of mining claims and the operation of mines of precious metals depended on these statutory enactments.

The first general legal work of the bureau was a complete collection, codification, and annotation of all congressional enactments relating to minerals, mineral lands, and mining. This work was completed in 1915. The results are printed in two large volumes containing in all 1,875 pages, entitled "United States Mining Statutes Annotated," and published as Bulletin 94. The work includes every enactment of Congress from the original ordinance of 1875 to the day of its publication and all sections of the Revised Statutes of the United States relating to the subject of mines. The statutes are grouped according to their general subject matter, and the statutes in each group are arranged in numerical or chronological order. In

addition to the sections of the Revised Statutes of 1878 and the acts relating to metal, coal, and oil and gas are included statutes bearing on the mining industry in Alaska, Indian lands, and the Philippine Islands and relating to lead mines, pipe lines, railroad grants, rights of way, salines and salt springs, settlers' relief acts, State and public grants, stone lands, timber cutting for mining purposes, town sites, tunnel acts, and withdrawals.

All sections of the Revised Statutes and all statutes at large not covered by them and statutes since enacted are thoroughly annotated. These annotations consist of carefully prepared abstracts of decisions of all courts and public offices in which any of these sections or statutes have been construed and applied.

The annotations are arranged under each section or statute and are grouped under prominent and appropriate title lines with a view to the logical sequence of the various subjects or titles and consist of plain provisions of law intelligible alike to miners and laymen. The annotations exhibit the present status of each particular section or act and its present-day application to the subject of mines and mineral lands. They show that the courts have cured obvious defects, made clear many uncertainties, and aided the practical application of the statutes in the matter of locating mining claims upon the public lands.

The decisions abstracted for the purpose of the work are found in the United States Supreme Court reports; the various Federal court reports; the decisions of the General Land Office as reported in various volumes; the decisions of the Attorney General; and the reports of the various metal mining States, including in all not less than 1,000 volumes.

It has been a matter of regret to the director that free distribution could not be made of this work to all persons requesting it; but the cost of publication for so large a bulletin required that it should be sold for a sum sufficient only to cover the actual expense of printing and binding, \$2.50.

CODIFICATION OF STATE MINING STATUTES.

The bureau is now engaged and has well under way a companion work proposed to include the mining statutes of every State arranged in chronological or logical order, with annotations following the plan and style of those in the United States mining statutes. This work will show the relative merits of the mining laws of the various States and is intended to aid in obtaining, so far as practical and possible, uniform mining laws.

Practical miners are more or less migratory. In passing from State to State they are confronted with different systems of laws and they must necessarily learn the leading features and know

more or less of the rules and regulations under which they work in the different States. It is believed that this is not only a cause of annoyance but is in fact a source of many accidents resulting in injury and loss of life. With a uniform system of laws throughout all the coal-mining States and a like system in the metal-mining States, miners may pass from one State and from one mine into another and ply their vocation as miners without embarrassment and with safer and saner results in their operations.

RULES AND REGULATIONS FOR METAL MINES.

Along this line of uniformity of mining laws and regulations the bureau takes pride in saying that in connection with the efforts of the American Mining Congress there have been worked out and given to the public uniform rules and regulations for metal mines. As early as 1906 the American Mining Congress appointed a committee consisting of W. R. Ingalls, chairman, J. Parke Channing, James Douglas, James R. Finley, and John Hays Hammond to draft a modern law governing quarrying and metalliferous mining which could be recommended to the several States for adoption, in the hope that the passage of such a uniform law would tend to lower the number of fatal and serious accidents. This committee, after the organization of the Bureau of Mines, worked in harmony with the bureau and has recently reported a draft of an ideal metalliferous mining act, including a general system of rules for the control of all metal mines, with general safety precautions. The bureau has the satisfaction of knowing that the rules and regulations thus recommended have been adopted by operators of some of the largest mines in the metal-mining States, and it is believed that sooner or later a substantially uniform law will be enacted by the several States.

IMPROVEMENTS IN COAL-MINING METHODS.

Efforts are being made in the different coal-mining districts to increase both safety and efficiency. In some districts progress has been more rapid than in others. However, the general interest in safety and the general desire to do everything possible to improve conditions is most encouraging.

WESTERN PENNSYLVANIA.

The room-and-pillar system of mining is in general use in this field. In some districts a panel system with about 25 rooms driven on each side of the pair of entries is usual. Most of the rooms are 20 to 24 feet wide and the pillars are 15 to 24 feet wide. In the system used most extensively rooms 280 feet long are driven off one side when advancing, the pillars being drawn as soon as rooms reach the

full distance, and 200-foot rooms are driven off the other entry in retreating. Under heavy cover there has been some difficulty in winning all the pillar coal on the retreat. In consequence, at some mines, the butt entries are 450 feet apart, the advance rooms are 350 feet long, and the 50-foot pillar on the other side of the pair of entries is taken when entry pillars are withdrawn. This modification has given greater success.

In the Pittsburgh district another development in mining methods is the use of a long-wall system of mining by a company not working the Pittsburgh coal. The coal is soft, the bed is about 3 to 4 feet thick, the cover is heavy, being 700 to 1,000 feet thick, and the roof breaks badly and often to a considerable height. The coal is mined from a 300-foot face and little shooting is necessary.

The adoption of miners' electric lamps, after a number had been approved by the Bureau of Mines, is proceeding rapidly. About 80,000 portable electric mine lamps approved by the bureau were in use in coal mines in the United States on January 1, 1917.

Shot firing by shot firers is being adopted extensively. In many mines assistant foremen act as shot firers. This practice gives the shot firer more authority, and should make for greater safety.

One large company uses permissible explosives exclusively, and another company uses them in about one-half of its gaseous mines. The general use of these explosives has been retarded somewhat by increased cost.

The coal-dust menace is generally combated by watering or the use of deliquescent salts. One company, in cooperation with the Bureau of Mines, has recently tried making the coal dust inert by the use of rock dust, with satisfactory results. The protection is much greater than by watering and the cost is not excessive. A wider adoption of dusting is probable if suitable grinding plants are constructed.

A large number of central rescue stations have been installed in western Pennsylvania, largely as a result of rating systems adopted by insurance companies and the educational work of the Bureau of Mines. Safety committees, frequent inspection, well-devised rules and regulations printed in several languages and distributed to the miners, abundant danger signals, mechanical safeguards for machinery, block signals for electric haulage, underground escape ways for the men in the larger mines using safety lamps, and well-organized first-aid and rescue teams are other features making for safety in the coal mines of western Pennsylvania.

Also, much work has been done toward improving the miner's living conditions by providing pure water, new dwellings, and better sanitation, and by encouraging the establishment of churches, schools, and playgrounds.

OKLAHOMA.

Since the beginning of coal mining in Oklahoma the general practice has been to shoot the coal off the solid. The mines are worked by the room-and-pillar method. Since the Secretary of the Interior issued orders covering mines on the segregated, or Indian, coal lands, there has been a decided trend toward better mining methods. In a few mines the advancing long-wall system, using machines, has been introduced with seeming success. With this method practically all the coal in the seam is recovered, in contrast to a recovery of only 50 to 60 per cent with the room-and-pillar mining. Various companies are introducing machines for undercutting the coal and are using permissible explosives for blasting. These improvements will undoubtedly work to the advantage of the industry.

A lack of electric power has handicapped the introduction of some improvements, but a central power company is now making arrangements for extending its lines to the various mines.

COLORADO AND WYOMING.

Shooting off the solid with black powder was an almost universal practice in Colorado and Wyoming until a few years ago. At present nearly all the larger companies mine the coal with machines or with hand picks, and in many mines permissible explosives are used. The room-and-pillar system is generally employed, modified by panel methods to meet differing conditions of cover, roof, and dip of bed. Most of the larger mines use electric current, each mine usually generating its own power. Both Colorado and Wyoming have been singularly free from coal-mine disasters in recent years, and this advance is largely attributed to better methods of mining.

Considerable attention is being paid to rendering coal dust harmless, both by humidifying and sprinkling, and by using rock dust. Open lights were formerly used in practically all mines, but now both electric lamps and flame safety lamps have been widely introduced.

Some of the larger coal companies have their own rescue cars, with attendants constantly at hand. During the past year much work has been done for the welfare of the miners. Clubhouses, washhouses, etc., have been introduced in many coal-mining communities. That measures for the welfare and safety of coal miners in Colorado are bearing fruit is indicated by the lessening number of fatal accidents. For the calendar year 1916 these amounted to 43, as compared with 63 for 1915, in spite of the fact that the coal output for 1916 was approximately 20 per cent more than for 1915.

SOUTHERN APPALACHIAN FIELDS.

The southern Appalachian fields include southwestern Virginia, eastern Kentucky and Tennessee, and northern Alabama and Georgia. Most of the mines in Alabama are opened by slopes, the room-and-pillar system is used, and the coal is shot off the solid with black powder. The use of panels has recently been introduced in some mines working pitching beds, the entries being driven along the strike of the seam, right and left from the slope, at intervals of about 1,000 feet, and at intervals of 1,000 feet along each level entry panel roads are driven up the dip. From the panel entries the rooms are turned at right angles parallel to the strike of the bed.

In Kentucky, Tennessee, and Virginia little coal is mined at great depths, and most mines are opened by drifts. The room-and-pillar system is used, the thickness of the pillars varying from a few feet to more than room width, according to the depth of cover. In some districts the pillars have been too thin, and parts of some of the mines have been lost through squeezes. The proportion of coal shot from the solid is annually becoming less and the number of mining machines is increasing.

Many of the mines are small and use mule haulage, but the larger mines use electric haulage. A few mines use tail rope.

Some mines still use furnaces for ventilation, but furnaces are gradually being replaced by fans of modern type.

Permissible explosives are being introduced gradually. Additional attention has been given to alleviating the coal-dust hazard, principally by means of water cars and watering systems. The increasing demand for better coal and the higher cost of production have caused many operators to consider improved mining machinery and methods, so that conditions in the industry as a whole show decided improvement.

CENTRAL INTERIOR FIELDS.

Operators in the central interior fields, including Indiana, Illinois, western Kentucky, Iowa, and northern Missouri, are tending to readjust their mine layouts to the panel system and to adopt mining methods that will increase the percentage of coal recovered. Storage-battery locomotives for gathering coal are being used more and more, and the introduction of the combination trolley and storage-battery locomotives has attracted considerable interest. Stripping operations in Illinois are very active, the latest plants using revolving steam shovels with 90-foot booms, the buckets having a capacity of 5 cubic yards. Further use of electric power generated by central stations is planned. Mines developing the deeper coal beds, especially in Indiana, are finding more gas, so that the necessity for giving closer attention to ventilation is evident. In the newer

mines the three-entry system is being adopted in preference to the two-entry system commonly used.

WEST VIRGINIA.

The year 1916 has been a period of great activity, the output of coal and coke has been large, and the general tendency has been toward the adoption of safer and more economical methods of mining. The topography of the coal-bearing area of the State is mostly rugged; hence some of the beds mined outcrop at 900 feet above a near-by stream, whereas others have shafts 600 feet or more deep. Room-and-pillar mining, with two, three, and four entry systems is used, the latter two being confined, as a rule, to large mines. So far as known, no long-wall mining is done in the State. Room-and-pillar mining with panels is especially adapted to many of the beds, on account of their being continuous, free from bad faults, and lying practically level.

Some of the most improved mine-ventilating plants in the United States are in West Virginia, many of the fan houses being of steel, stone, or concrete. As a result, the fire hazard at fans has been reduced to a minimum. In most of the recent plants the fans are not placed in direct line with the main airways. Stoppings of noncombustible material are required on all main airways. Rope haulage is used to some extent, but mechanical haulage is chiefly through electric locomotives. Steam, gasoline, and compressed-air locomotives are also used. Mule haulage is generally confined to gathering the coal. There are many small, isolated power plants, but centralized power stations are gaining favor. Many mines are served by hydroelectric plants.

In order to prevent accumulations of coal dust underground, the general custom is to remove loose coal and dust as fast as possible. Sprinkling systems are installed in a great many mines, and various methods of humidifying the mine air are practiced. In mines that generate explosive gas, permissible explosives only are used. Except where permission has been granted by the district mine inspector, shooting off the solid is expressly forbidden. Permission is not granted where conditions favor, even remotely, the initiation of an explosion by solid shooting. Shot firers are employed in gaseous mines. The use of coal dust or other inflammable material for stemming is forbidden by law.

In 1897 there were only 16 machines in use, whereas to-day more than 55 per cent of the total output of coal is won by mining machines.

First-aid and mine rescue training was conducted in the State by the Bureau of Mines during seven months in the fiscal year ended June 30, 1916. The first-aid work in particular was received with enthusiasm both by the miner and the operator.

ANTHRACITE DISTRICT OF PENNSYLVANIA.

Mining methods in the anthracite district are governed by the dip of the beds, their depth and thickness, and the character of the roof and floor. Where the coal is comparatively flat, ordinary room-and-pillar mining is employed, and the double-entry system is generally used. Considerable attention has been given to columnizing pillars, with reference to those of workings in overlying or underlying beds, with marked success. In heavily pitching seams, where the coal is not too thick, the so-called battery method is employed. In practically all the field the hazards of mining are increased by the presence of fire damp. Because of gas, the great extent of the mines, and the many rolls, folds, and faults, careful attention is given to ventilation. Fans are used almost exclusively. Most of the fans are of the Capell type, large diameter, and relatively low speed, but in the past 15 years many small-diameter high-speed fans have been installed.

There are three general types of haulage employed—mule, rope, and motor. Mules are used largely for gathering the loaded cars from the chambers. On account of the large number of slopes and planes rope haulage is much employed. During the past few years, however, the electric motor has come into wide use, both for gathering and for main haulage work.

In most mines it is not practicable to undercut the coal before blasting, so that shooting off the solid is the general practice. However, machine mining has been introduced in horizontal beds at some mines within the past few years.

As regards safety and health, the chief improvement in mining practice in recent years has been the introduction of better lights. Carbide lamps have largely superseded oil-fed torches in open-light mines, and many mines have adopted approved flame safety lamps. Large numbers of electric cap lamps are in use and have proved so satisfactory that more will undoubtedly be used.

Practically every mine in this field has a first-aid team. Both miners and operators show great interest in first aid, and it is probable that hundreds of men each year owe their lives to the efficient first-aid service. Several of the larger companies maintain fully equipped cars for rescue work, fire fighting, and hospital purposes, and there are many privately owned rescue stations in charge of specially trained men.

Living conditions have probably always been better in the anthracite field than in most coal-mining districts, but have greatly improved in recent years. Progress in bettering conditions that bear on health, safety, and efficiency continues, and the outlook for further improvement is particularly bright.

FUELS AND MECHANICAL-EQUIPMENT INVESTIGATIONS.

IMPORTANCE OF INVESTIGATION OF FUELS.

Last year the people of the United States burned some 500,000,000 tons of coal. If this coal had been used with the highest efficiency, it is probable that some 125,000,000 tons, or 25 per cent, would have been saved. Such figures convey an idea of the possible saving to the people of this country through the proper preparation and handling of coal and through the development and utilization of efficient coal-burning equipment. With a view to realizing greater efficiency in the preparation, treatment, and use of the fuels burned in this country, Congress has appropriated funds for investigations to be made by the Bureau of Mines and for the publication of the results of these investigations.

CHARACTER OF INVESTIGATIONS.

Such work naturally divides itself into two general investigations, one having as its object the raising of average efficiency in the general use of fuels and the other relating directly to Government plants. As the results of the general investigations may be applied to the power plants of the Government, tests of a fundamental character are made with the object of clearing up misapprehensions as to the nature and properties of the many available fuels and as to the principles of combustion and heat generation and the transmission of heat to any desired place.

During the past 10 years there has been a much closer scrutiny of the combustion process in all kinds of fuel-burning equipment. For a long time engineers concentrated attention upon the improvement of prime movers—the steam engine, the gas engine, and other internal-combustion engines—the burning of coal under boilers and in gas producers being somewhat neglected. Technical training in our colleges placed relatively small emphasis on the principles and the process of combustion, as compared with a study of the details of prime movers. The more evident facts of combustion seemed so simple as to offer little inducement for an intensive study of the process, but, as a matter of fact, the burning of a fuel is by no means

a simple process, and the misconceptions that have resulted from a too casual study of the phenomena have led engineers to construct uneconomical and ineffective devices. The clearing away of these misconceptions can come only by knowledge of the actual facts, and for that reason the Bureau of Mines has been making a careful study of the combustion process.

The nature of coal has also been misunderstood by many users. Coal is an extremely complex substance, behaving quite differently under different conditions of combustion, and as the United States is extremely rich in a great variety of mineral fuels, it would seem necessary that furnace practice and design should vary with the characteristics of the fuels used. As a matter of fact, the variety of coal-burning equipment now existing is the result of ill-advised experimentation rather than rational design, and the engineer has been unable to find quantitative figures that would enable him to design a furnace for a given fuel with the same assurance of success that he has in the design of a bridge or other structure. Studies of this intimate kind are usually beyond the possibility of private investigation, but are believed to be most essential to the efficient use of the fuel resources of the country.

The attempt to decrease the cost of power has led to the development of internal-combustion engines using either gas or liquid fuel, and these engines have a much higher efficiency than the usual steam engine. However, they also use a much more expensive fuel, so that their use is confined to certain relatively narrow fields. One characteristic of these engines is that their high efficiency varies through a relatively narrow range whether they are large units or small units, whereas the economy of the steam engine varies through a wide range, and the most efficient types have efficiencies comparable with those of many internal-combustion engines.

The possibilities of the more efficient use of steam machinery are therefore very large. Large turbo-generating units and efficient boilers are now available that return as useful electric current ready for distribution nearly 20 per cent of the potential energy in the coal. Although the average efficiency of all kinds of steam-power plants in the United States can be only a matter of guess, it is quite probable that the average is somewhere in the neighborhood of 5 or 6 per cent of the energy of the coal transformed into useful energy ready for distribution; so that if it were possible to elevate the average efficiency to something near the maximum now attainable in steam plants, about three times as much energy would be available for the productive industries of the country as is now obtainable from the coal burned. Anything that can be done, therefore, to raise the general average of efficiency by the production of more efficient furnaces, heat-absorbing devices, and steam engines will be in the line of con-

structive conservation of our natural resources. The work of the Bureau of Mines in its fuel investigations and in its study of heat transmission is directed toward increasing this efficiency.

VARIETY OF FUELS AVAILABLE.

The adherence to the use of certain fuels is many times largely a matter of habit, and as the favorite fuel becomes more and more expensive a change to other forms of fuel is made with considerable difficulty. Looked at in a broad way it is not economical to transport high-grade coal through long distances into districts where fuels of lesser value are to be had in abundance, when, as a matter of fact, the cheaper fuels could be satisfactorily used if the user would show a little patience and ingenuity. The suitability of fuels for use in any locality is therefore a matter of public education in which the Bureau of Mines takes a deep interest. As an illustration, the substitution of coke for anthracite coal in many localities is desirable on the score of economy, and the bureau desires to stimulate the use of coke as a domestic fuel because of its cleanliness. The bureau is also endeavoring to stimulate the use of coal gas, another form of smokeless fuel, instead of water gas, because during the past year some 20,000,000 barrels of petroleum was required to enrich the water gas used. The price of petroleum has so increased and the demands on our petroleum resources are so great as to make the use of this product undesirable for services that can be performed by other fuels.

New methods of using fuel are also being proposed, and the bureau desires to be informed as to the technical success of these methods in order to give correct information and thus conserve not only capital but also the fuels by quickly finding the special field particularly adapted to each new method. This function was well illustrated in the introduction of the gas producer, which not many years ago was a new device in the United States and concerning which the Bureau of Mines gave information in the early stages of development. This device has already found its field of usefulness, and newer methods of using coal are now being presented. Just at present there is great interest in powdered fuel, which will doubtless find its special field of usefulness.

GOVERNMENT FUEL PROBLEMS.

Another field of work for the bureau is in the application of good engineering to the fuel problems of the Government itself. The Government fuel bill is somewhere in the neighborhood of \$8,000,000 per year. Some of this fuel is burned under efficient conditions that make the practice well above the average of the country, but many of the Government plants are antiquated and have suffered from

lack of a consistent engineering policy. For these plants the Bureau of Mines acts in the capacity of a consulting engineer in fuel problems having to do with the purchase of the most suitable fuel that the particular market affords and in recommending practices and equipment for the more efficient use of such fuel. It has been found advantageous to have engineering advice on these special problems from a source not too intimately connected with the local administration of a particular plant, and as the bureau is making a special study of such fuel problems it has been able to render service to Government plants scattered all over the country.

IMPORTANCE OF INVESTIGATIONS OF MINING EQUIPMENT.

The safe and economical conduct of mining operations depends very largely upon suitable mechanical equipment. New devices are continually being proposed, some having elements of danger not at all apparent. When a good device has been tried and dropped because of some unforeseen limitation as to safety the art suffers a considerable loss, and in the beginning of the development of new appliances it is highly desirable that there be careful investigation and that the development be directed toward safety.

This need applies peculiarly to electrical devices, and for that reason the Bureau of Mines is especially directed to investigate the use of electricity in mines. The great flexibility and ease of transmission of electric power, and the relatively cheap cost of electrical equipment has led to the wide use of electrical devices for coal cutting, transportation, and hoisting. However, some of the equipment is so designed as to increase the dangers of mining. A large part of this danger can be eliminated by suitable design and proper oversight and operation. Electrical devices are therefore carefully investigated by the Bureau of Mines, with the object of aiding development along safe lines. For this purpose the bureau issues an approval of any device that meets the specifications prepared by the bureau. This approval system applies particularly to miners' electric lamps, to explosion-proof motors, switches, and coal-cutting machines. In addition, motors for general service and storage-battery electric locomotives are under investigation by the bureau.

Old and well-known equipment is frequently the source of potential danger. Through its long use many of the dangers have become thoroughly known, but there are some not so evident that will yield their secrets only to long and careful special investigation, and for that purpose the bureau is providing, at its Pittsburgh plant, laboratories for the investigation of mining equipment. These laboratories are in course of construction and will be occupied during the coming year.

BOILER-FURNACE INVESTIGATIONS.**CONSUMPTION OF COAL IN UNITED STATES.**

As has been mentioned above, the various branches of industry in the United States consume annually over 500,000,000 tons of coal. If the average price of coal at the place of consumption be assumed to be not less than \$2 per ton, the total value of the coal consumed annually in the United States is considerably over \$1,000,000,000. The Federal Government alone annually uses coal to the value of about \$10,000,000. This huge quantity of coal is used for various purposes. However, the largest part of it, amounting to considerably over one-half, is used for power production in large central stations, small isolated power plants, locomotives, steamboats, and many manufacturing plants. Smaller quantities, that amount to millions of tons in all, are used for metallurgical and chemical purposes, and for the production of coke.

In all these uses of coal the efficiency of the processes that are employed is low, and seemingly there are good chances for improvement. Thus, for example, in the most efficient large power plant of to-day scarcely 20 per cent of the heat in the coal consumed is converted into mechanical power, and in the small power stations the efficiency frequently drops below 10 per cent. In the process of producing coke in the beehive coke ovens, most of the volatile matter, representing about 30 per cent of the total heat in coal, is discharged into the atmosphere. Thus the coking process not only wastes the volatile matter but contaminates the atmosphere for miles with poisonous fumes. Surely when such a large sum of money is expended annually in clearly wasteful processes, investigation by the Federal Government into the possibilities of reducing the wastes by improving the processes is not only desirable but seems imperative. The expenditure for the coal is so large that the saving of even a small percentage means the saving of large sums of money. If any investigation should result in a saving of only 0.1 per cent, such saving would still amount to \$1,000,000 annually, a sum that would cover the expense of an extensive investigation. Usually a much larger saving than 0.1 per cent can be brought about by merely calling the consumers' attention to the large wastes attending the different processes.

By properly conducted tests and research work, more efficient processes can be substituted, and a saving measured not in fractions but in whole numbers can be accomplished.

TWO GROUPS OF BOILER-FURNACE INVESTIGATIONS.

The boiler-furnace investigations of the Bureau of Mines comprise both field and laboratory investigations.

The field investigations are made at the different plants belonging to the Government and at plants belonging to private companies cooperating with the bureau. These investigations are made under actual operating conditions. Their object is mainly to determine the efficiency of the processes used in the plants and the possible methods of reducing the wastes.

The laboratory investigations consist of such research work as can not be done with commercial apparatus under actual operating conditions and for which special apparatus must be designed. Laboratory investigations with special apparatus are necessary because of the nature of the many processes. The result of such processes are affected by several factors which vary independently in the actual operation of commercial apparatus, so that it is impossible to determine the effect of any one factor. It then becomes necessary to design a special apparatus in which all the factors except the one under observation can be kept constant. Thus, for an example, an investigation of the effect of the diameter of the tubes in a locomotive boiler on the efficiency of the boiler may be cited. If such an investigation were made with an ordinary hand-fired locomotive, by using successively sets of tubes of different diameters and running tests with each set of tubes, the results of the test would show not only the effect of the diameter of the tubes, but also the effect of the construction and the performance of the furnace, the effect of variation in coal, the effect of the skill of the fireman, and the effect of many other minor factors.

Under such testing conditions the effects of the various factors would be so intermingled that it would be decidedly difficult to tell with any accuracy the effect of the size of the boiler tubes on the efficiency of the locomotive; moreover, such tests would be expensive. To get accurate and reliable data the disturbing effects of all the factors except the size of the tubes must be eliminated. To eliminate the effect of the condition of the road, of the engine, and of the engineer the tests can be made only when the locomotive is standing in the railroad yards. To eliminate the effect of nonuniform coal fuel the boiler can be fired with natural gas of uniform quality; to eliminate the fireman and the variation in furnace conditions the gas and the air can be fed into the furnace at a uniform rate, the temperature of the products of combustion and their velocity through the boiler tube being thus kept constant, so that only the effect of the variation of the size of the tube remains.

Similar methods of elimination of undesirable variables are applied to other problems in laboratory investigations. Many people object to laboratory investigations on the ground that they are made with small apparatus, but such investigations need not be limited to small-size apparatus. A large apparatus can be used as well. The matter

of size usually depends on the funds available for such investigations. If the funds are sufficient an apparatus as large as commercial equipment can generally be used. The main requisite for the successful investigation of any laboratory apparatus is that all conditions can be easily controlled so that any number of the factors can be kept constant and any single factor can be varied through a desirable range.

LABORATORY INVESTIGATIONS OF BOILER FURNACES.

In studying the utilization of fuels there are mainly two processes to be considered, namely, the production of heat by the combustion of the fuel and the application to industrial processes of the heat produced. Thus a large part of the problem of efficient fuel utilization resolves itself naturally into the investigation of the combustion of fuels and the investigation of the transfer of heat from the products of combustion to where the heat is needed. For these reasons the work of the bureau's laboratory tests of fuel has been along two main lines—studies of combustion and studies of heat transmission.

OBJECT OF COMBUSTION INVESTIGATIONS.

The general object of the combustion investigations is to obtain definite information on the combustion process and thus enable furnace designers to determine the proportions of a furnace to burn economically any given fuel under any given conditions. The information heretofore available on the combustion process is not definite enough to permit rational furnace design, or, at least, is not as definite as the information available to the designer of machinery. Thus, for instance, the tensile strength of steel may be known to be 60,000 pounds per square inch of cross section, and when a machine part that will be subjected to a given tensile stress is being designed, the part can easily be proportioned to withstand the working stress with any desirable factor of safety. When, however, a furnace to burn a given coal at a given rate to any desirable degree of completeness is to be designed, only meager information is available. Such data as are at hand are rather qualitative than quantitative; that is, the data are mostly given in such terms as "large" or "small" or "short" and not in so many feet or so many pounds. As a result of this lack of definiteness one designer will design a furnace with 3 cubic feet of combustion space to each square foot of grate area, whereas another designer makes the same ratio 8 to 1, both furnaces being intended to burn the same coal at the same rate. We know that a furnace to burn Illinois coal must have a larger combustion space than a furnace of similar design to burn Pocahontas coal, but we do not know what the exact figures expressing the difference should be.

We also know that to burn 1 pound of coal we must supply 14 to 20 pounds of air, some of this air being supplied through the grate and the rest over the fuel bed, but we do not know what proportion must come through the grate and how much of it should be introduced over the fuel bed. As a result of this absence of accurate data many furnaces are in use that have only inadequate provision for the introduction of air over the fuel bed, and some that have no such provision, the air supply depending on the accidental admission of air through the holes in the fire or on leakage around the firing door and through cracks in the wall. These examples illustrate the lack of definiteness of the information now available for a furnace designer, and show the desirability of combustion investigations.

The bureau's fuel-efficiency laboratory is studying the process of combustion in the fuel bed and also in the combustion space of coal-burning furnaces, and is determining some of the quantitative data mentioned above.

COMBUSTION IN FURNACE FUEL BED.

Combustion in the fuel bed is studied by means of a small vertical furnace having 1 square foot of grate area. The air admitted through the grate is supplied under pressure and is measured with an orifice. Provisions are made for taking gas samples and measuring temperatures in the fuel bed at intervals of $1\frac{1}{2}$ inches from the grate. Thus, the composition of the gases and the temperature at the various heights show the process of combustion in the fuel bed. The small furnace was selected for this study in order to make possible accurate gas sampling and temperature measuring in the fuel bed. The grate area was nearly as large as in a small house-heating boiler. Supplementary work with large furnaces confirmed the results obtained with the small furnace.

EFFECT OF CLINKERING.

The same laboratory furnace is used for investigating the relation between the fusibility of coal ash and the clinkering of coal in furnaces. Some fuel men have argued that the value of a coal as fuel depends directly on the fusibility of its ash, the lower the softening point of its ash, the less its value as a fuel for steaming purposes. Also, some fuel experts have insisted that the melting point of the ash should be embodied in the specifications for buying coal, saying that it is as important as the heating-unit determination, and a few men went so far as to advise disregarding the heating-unit determination and advocated the buying of coal entirely on the basis of the softening point of the ash. The fact of the matter is, however, that it is by no means settled that as the softening point of

ash drops, clinker troubles increase. When making tests at Government plants one of the bureau's engineers noticed on several occasions that coal giving trouble from clinker had ash of a higher softening point than another coal that did not give any trouble from clinker. Consequently the cause of clinker trouble appears to be more complex than would at first appear, and it would seem that the time has not yet arrived for placing the softening point of ash into coal specifications, because a great harm could be done unjustly to many coal-mine operators.

It is for these reasons that the bureau is studying the relation between the softening points of coal ash and the clinker trouble. The small furnace has been selected because the same furnace conditions can be easily reproduced for all the coals that may be subjected to the study.

COMBUSTION-SPACE INVESTIGATIONS.

The process of combustion in the combustion space of a furnace is studied with a furnace having 30 square feet of grate area and a combustion space 3 feet by 3 feet in cross section and about 40 feet long. Samples of gases are collected at intervals of 1 to 5 feet along the path of gases through the combustion space. Temperatures are also measured at the same cross sections. The composition of the gases and the temperature indicate the progress of combustion in the space. A detailed description of the combustion investigations and the results are given in Bureau of Mines Technical Papers 63^a and 137^b.

PROCESS OF COMBUSTION.

The process of combustion, beginning at the surface of the grate and progressing through the combustion space, is shown in figure 4. The curves represent the variation of the percentage of the different gases and indicate the process of combustion. Thus the atmosphere at the surface of the grate contains 21 per cent oxygen—that is, the full proportion in the air. As the air passes through the lower layers of the fuel bed the oxygen is rapidly used in the combustion of the coke and entirely disappears 3 to 4 inches from the grate. The carbon dioxide content is zero at the top of the grate but quickly rises and reaches a maximum of about 12 per cent 3 to 4 inches above the grate—that is, at the same point where the oxygen disappears. Beyond this point the carbon dioxide decreases. The combustible gas, consisting mostly of carbon monoxide, starts with zero at the

^a Clement, J. K., Frazer, J. W. C., and Augustine, C. E., *Factors governing the combustion of coal in boiler furnaces*: Tech. Paper 63, Bureau of Mines, 1914, 46 pp., 26 figs.

^b Kreislinger, Henry, Ovitiz, F. K., and Augustine, C. E., *Combustion in the fuel bed of hand-fired furnaces*: Tech. Paper 137, Bureau of Mines. In press.

grate and rises slowly at first, but at the point where the oxygen disappears it rises rapidly. After the oxygen disappears the carbon dioxide acts as an oxidizing agent in the combustion of the coke and

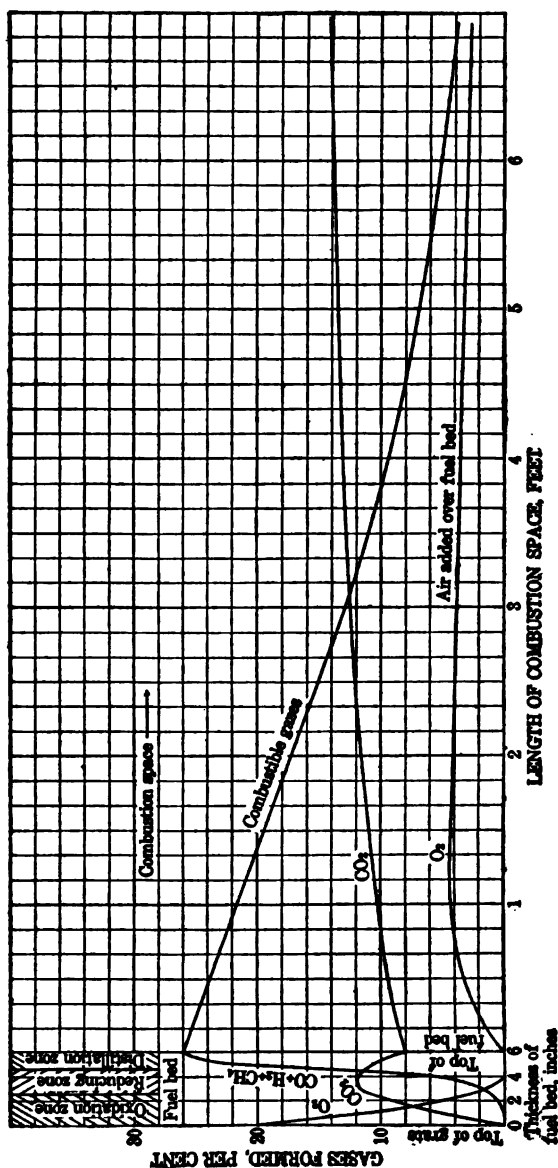


FIGURE 4.—Composition of gases in the fuel bed and in the combustion space. The gases at the top of the fuel bed contain no free oxygen and a high percentage of combustible gases, which burn in the space beyond the fuel bed after oxygen has been admitted. This diagram is a generalization of the results of many tests.

is itself reduced to carbon monoxide. To this carbon monoxide thus formed is added in the layer at the top of the fuel bed the volatile combustible distilled from the freshly fired coal, bringing the total combustible gas rising from the fuel bed to 82 per cent.

Thus in the hand-fired furnace, or in any furnace into which coal is fed from the top, there are three zones of different reactions. In the lowest $2\frac{1}{2}$ or 3 inches the coal is oxidized to carbon dioxide, and this layer is therefore called the oxidizing zone. In the layers above the oxidizing zone the carbon dioxide is reduced by contact with hot coke to carbon monoxide; therefore, the layer is called the reducing zone. In the layer at the surface of the fuel bed the volatile matter is distilled from the fresh coal, and therefore this layer is called the distillation zone. These different zones are not sharply separated, but merge into one another, particularly the reducing zone, which probably extends through the distillation zone. These different zones of the fuel bed are shown at the top of figure 4.

Thus far the combustion in the fuel bed had been considered. The gases rising from the fuel bed contain 20 to 32 per cent combustible matter, 5 to 8 per cent carbon dioxide, and practically no free oxygen. Besides the combustible gases there are soot and some tars, the latter decomposing quickly into soot and gases. Air must be added to all of these combustible materials rising from the fuel bed, and the combustible materials must be mixed with the air and burned in the combustion space beyond the fuel bed. Thus in the combustion space occur two processes, namely, mixing and burning. The changes taking place in the combustion space are indicated in figure 4 by the curves extending beyond the fuel bed. The figure shows that the oxygen above the fuel bed quickly rises, the combustible matter drops, and the carbon dioxide increases about at the same rate as the combustible matter drops. The combustible matter approaches the zero line as the process of combustion nears its completion. It should be noted that the process of combustion in the combustion space is not as rapid as it is in the fuel bed, and that to obtain nearly complete combustion in any given furnace a considerable volume of combustion space must be available. As the combustion space determines the completeness of the combustion of coal, and therefore to a large extent also the economy in the utilization of coal, the correct relation of the size of combustion space to the completeness of combustion for any given coal, burning under any given set of conditions, is of much importance in the art of furnace design.

SUMMARY OF DEDUCTIONS.

Following is a summary of deductions from the investigations so far conducted by the Bureau of Mines:

The fuel bed in almost any coal-burning furnace acts primarily as a gas producer. In a hand-fired furnace when the fuel bed is level and 5 inches or more thick, the gases rising from the bed contain 25 to 32 per cent of combustible gases, about 5 to 8 per cent of CO_2 ,

and no free oxygen; in other words, the gases constitute a fairly good producer gas.

Only about $6\frac{1}{2}$ pounds of air per pound of coal can be forced through the fuel bed, no matter how fast the air is blown through. When the quantity of air thus supplied is doubled the rate of combustion is doubled, when the quantity of air is quadrupled the combustion is quadrupled also, and so on, the weight of air per pound of coal burned remaining nearly constant at $6\frac{1}{2}$. This means that the rate of combustion depends on the rate of air supply through the grate. This relation is true for all fuels, including anthracite and coke.

When the fuel bed is level and 5 inches thick at least one-half of the air required for the complete combustion of the coal must be supplied above the fuel bed. This air should be supplied as near to the fuel bed as practicable, and should be introduced in many small streams at high velocity in order to facilitate mixing. The combustible gases rising from the fuel bed represent 40 to 60 per cent of the total heat in coal, so that it can be said that on the average one-half of the combustion of coal takes place in the fuel bed and one-half in the combustion space.

The completeness of combustion of the gases depends largely on the size of the combustion space. Coals having different composition require different sizes of combustion space for the same completeness and the same rate of combustion. Roughly speaking, the combustion space should be proportioned to the product of the percentage of the volatile matter multiplied by the quality of the volatile matter, the ratio of the volatile carbon to the available hydrogen being taken as an indicator of the quality of the volatile matter. The size of the combustion space is also approximately proportional to the percentage of oxygen in moisture-free and ash-free coal.

The percentage of excess of air giving the best results in any steam-generating apparatus varies with the size of the combustion space and the kind of coal. Of two furnaces burning the same fuel but having different sizes of combustion space, the one with the larger combustion space gives the best results with lower excess of air than the one having the combustion space smaller. Similarly of two furnaces exactly alike in size, but burning different coals, the one burning the coal lower in volatile matter and oxygen gives the best results with lower excess of air than the furnace burning the coal higher in volatile matter and oxygen.

The volatile matter leaves the fuel bed as complex hydrocarbon compounds which at atmospheric pressure and temperature are in a liquid or semiliquid state. In the absence of sufficient oxygen for their complete combustion, the tars are quickly decomposed by the high temperature into soot and light gases such as hydrogen and car-

bon monoxide. The formation of carbon monoxide is due to the presence of carbon dioxide and a small supply of oxygen. At a distance of 1 or 2 feet from the surface of the fuel bed only a small amount of hydrocarbons can be found in any state, gaseous, liquid, or solid. The solid substance present in the flames is mostly soot with only a trace of tars. All hydrocarbons are unstable at high furnace temperature and unless a sufficient air supply is quickly mixed with them at the time they are distilled to insure their complete combustion, they are quickly decomposed, causing the deposition of soot. It is therefore useless to search for hydrocarbons several feet from the fuel bed. Methane, which is perhaps the most stable hydrocarbon, is found only in traces 1 foot from the surface of the fuel bed. The persistence of carbon monoxide in furnace gases is not due to the difficulty of burning it, but to its constant formation by the reaction between soot and carbon dioxide. This is the reason why carbon monoxide is found in the furnace gases after all other forms of combustible have practically disappeared.

Soot, which is the principal constituent of smoke, is formed at the surface of the fuel bed by the heating of the hydrocarbons in the absence of air. It is not formed by the hydrocarbons striking the cooling surfaces of the boiler. As a matter of fact, under ordinary conditions of operation only a small trace of the hydrocarbons ever reaches the surfaces of the boiler. Any hydrocarbon that reaches the surface of the boiler is kept from decomposing by the cooling effect. The cooling surfaces do not cause the formation of soot; they merely collect the soot and prevent its combustion.

Soot or smoke is formed at the surface of the fuel bed by the absence of oxygen. Therefore, to prevent soot sufficient air should be mixed with the volatile matter at the time of distillation. Any air added later is usually added too late. In other words, to obtain smokeless combustion, the distillation of the volatile matter must take place in a strongly oxidizing atmosphere. This is one of the main reasons of the success of most mechanical stokers in burning smoky coals without smoke.

HEAT-TRANSMISSION INVESTIGATIONS.

Inasmuch as the largest part of the coal consumed in the United States is used for making steam, the heat-transmission investigations of the Bureau of Mines fuel-efficiency laboratory are planned and conducted with especial reference to steam boilers. The object of the investigations is to obtain data that will enable engineers to design more efficient boilers. Once the boiler is designed, made, and set up, little can be done by the operator to improve its efficiency as a heat absorber. The operator can exercise more care in burning the coal with the most economical air supply, thereby making more heat

available for the boiler, but he can do little to make the boiler absorb a larger percentage of the heat available for absorption. In the past the design of steam boilers was considered in regard to mechanical strength and not the efficiency of heat absorption. Thus, fire-tube boilers were made with tubes 6, 4, 3, and, in locomotives, even 2 inches in diameter, seemingly no thought being given to the effect of the size of the tubes on the efficiency of the boilers. The idea seems to have prevailed that the amount of surface was the main factor in the efficiency of a boiler and that the arrangement of the heating surface had little to do with it, that is, that a 4-inch tube would absorb as much heat as a 2-inch tube as long as the two tubes had the same amount of heating surface, and as long as the same weights of hot gases at the same initial temperatures were forced through them. About nine years ago the Government's experts showed boiler tubes having a small diameter were more efficient than boilers with tubes of large diameter, but the exact relation between commercial sizes of tubes is not known. This exact relation between the size of tubes and boiler efficiency is one of many other factors now being determined by the bureau's fuel-efficiency laboratory.

The effect of boiler scale on the boiler efficiency is another important problem confronting the steam-boiler operator. A great deal has been written about it, but no really authoritative data are available to substantiate certain statements that are frequently made in engineering literature. This problem is another part of the heat-transmission investigations.

The bureau's heat-transmission investigations are made principally with an apparatus especially designed for that purpose. The apparatus consists mainly of a horizontal tubular boiler having a shell 20 feet long and 30 inches in diameter. Different sets of tubes ranging from 1 to 4 inches in diameter are being used in this boiler. The length of the boiler and of the tubes is being varied from 20 to 5 feet in steps of 5 feet.

The boiler is fed with the products of combustion from burning natural gas. The volume of the gas and that of the air used in combustion are accurately measured and controlled. The temperatures of the products of combustion entering and leaving the boiler are also measured. The control of the temperature is such that it can be kept constant at 1,000° or 1,200° C. within less than 10° C. The velocity of the gases through the boiler can be kept constant within about 5 per cent. All the factors during a test can be so controlled that closely accurate data can be obtained.

Matters already investigated or to be studied are as follows: Initial temperature of products of combustion; velocity of products of combustion through boiler; diameter of tubes; length of tubes; thickness of wall of tubes; pressure of steam in boiler; retarders in

tubes; boiler scale on the water side of tubes; soot on the gas side of tubes.

Besides the work outlined above, many special observations are made in connection with tests with this apparatus and in connection

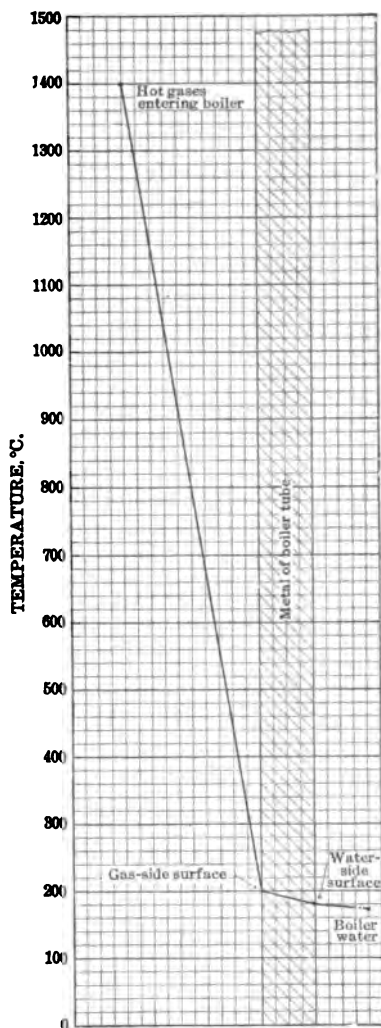


FIGURE 5.—Temperature drop along the path of heat travel from the hot gases into the boiler water. In the diagram the horizontal distances have no significance.

with tests of commercial boilers. These observations furnish much valuable information. Thus, a series of experiments was made on the accuracy of the measurement of temperatures of gases in boiler tubes and boiler settings. The temperature of the metal of the different parts of a boiler was determined while the boiler was in operation, a determination that requires the development of new methods of doing the work.

SUMMARY OF RESULTS.

Following is a summary of the results so far obtained from the heat-transmission investigations.

The path of the heat travel from the hot products of combustion to the boiler water consists of three distinct parts, namely, from the hot products of combustion to the dry surface of the heating plate; from the dry surface through the metal of the plate to the wet surface of the plate; and from the wet surface to the body of the boiler water. Of these three parts of the path the first part is by far the slowest, and is the one responsible for the slow rate of heat transmission in steam boilers. This fact is shown strikingly by figure 5, which indicates the temperature drop along the path of heat travel

in a Heine boiler at the place where the hot gases enter among the tubes when the boiler is worked at about 30 per cent above its rated capacity. The temperatures were obtained by actual measurements. The figures show that by far the greatest temperature drop is from the hot gases to the gas-side surface of the boiler tube.

Inasmuch as the resistance to the heat travel is proportional to the temperature drop, the resistance through the metal of the tube into the boiler water is insignificant. Therefore, anything that increases the rate of heat transmission from the hot gases to the dry surface of the heating plate almost directly increases the rate of heat transmission into the boiler water. It appears that the heating plate of a boiler can transmit any amount of heat that can ever be imparted to it by the hot gases.

The rate of heat impartation by the hot gases to the heating plate of the boiler is closely proportional to the temperature difference between the gases and the dry surface of the heating plate; to the velocity of the gases flowing over the surface; and to the density of the gases. As the temperature can not be raised beyond a certain limit, and as raising the temperature reduces the density of the gases, the velocity is the most important factor in increasing the rate of heat impartation from the hot gases to the heating plate of the boiler. Therefore, with any given temperature the capacity of the boiler can be increased at will by increasing the velocity of the gases.

The efficiency of a boiler as a heat absorber can be increased by reducing the diameter of tubes in fire-tube boilers and by placing the tubes closer together in water-tube boilers. By so doing the unit gas passages are made smaller in cross section so that the average distance of the particles of hot gases to the heating plate is shorter, and therefore heat is imparted to the plate at a faster rate. In order not to reduce the total cross section of the gas passage the number of unit passages can be increased. Thus, in a fire-tube boiler the desirable results can be obtained by using a larger number of smaller tubes.

The measured temperatures of the gases inside the boiler setting are usually too low on account of heat being radiated from the measuring instrument to the surrounding boiler surfaces. The larger the diameter of the instrument, the larger is the error. The best results are obtained when an exposed thermocouple of very fine wires is used. The hot junction should not have a bead but should be of nearly the same diameter as the wire itself. The measurements obtained with large thermocouples may be in error by hundreds of degrees.

FUEL-EFFICIENCY INVESTIGATIONS.

The selection of fuel for Government plants is often governed more by considerations as to its suitability under the existing conditions than by the comparison of guaranteed heating value or ash content. The Bureau of Mines conducts many tests for the purpose of obtaining information as to the suitability of various fuels for use in different kinds of equipment. On the basis of the information obtained it is enabled to give advice to other Government bureaus or depart-

ments as to the most desirable fuel to be purchased for use in the equipment available. This same information is also of use in making recommendations for the selection of new equipment for installation in Government plants, with the end in view that the new equipment may be suited to the greatest possible number of coals in the local market, especially the cheaper coals.

The suitability of any particular fuel may be determined by its tendency to cause smoke emission or to produce objectionable clinkers under some conditions, or by other factors. Frequently, however, a fuel considered unsatisfactory by the operating force in a given plant may, when differently handled, prove to be suitable. This statement applies to power plants or heating plants of considerable size, but is just as true when applied to residence heating conditions and equipment. Consequently the bureau frequently has to conduct service tests with some of the coals offered for a Government power or heating plant before recommending which fuel shall be used.

During the past year tests of this sort were carried out in four power plants in Washington, D. C., and also at two of the United States Indian schools. The coal for most Government plants is purchased on a specification basis, and the practice is to purchase the fuel showing the lowest guaranteed cost per million British thermal units, providing it is suitable for the plant conditions. At two other Indian schools new power plants or extensions to existing plants were to be made, and plans and specifications for this new work were examined by the bureau. In each case changes were recommended so as to make the equipment operate more efficiently and at the same time make it suited to the largest number of the fuels available.

A similar service is being rendered to the superintendent of the Government Hospital for the Insane in Washington, where acceptance tests and inspections of new stoker and coal and ash-handling equipment are being made.

Requests are also made of the bureau to make examinations of power-plant conditions and to indicate wherein these conditions can be improved or better results obtained. A case of this sort that occurred during the past year was that of the District pumping station, at Washington, D. C. The inquiry for this plant covered not only possibilities of greater economy in the utilization of fuel and steam, but also included the question as to the advisability of a change in fuel and in stoking equipment. In order to answer the inquiry it was necessary to conduct evaporative tests and to study the distribution of the steam used.

In addition to the tests relative to the selection and use of fuel carried on at other Government plants, requests are made by different departments of the Government for information obtained

from tests conducted at the bureau's experiment station in Pittsburgh. During the year 1916 several such problems have received attention. A series of tests to determine the comparative evaporative performance of two types of heating boiler such as would be used in post-office buildings were carried out for the Treasury Department. The purpose of this study was to ascertain whether one type would show fuel savings sufficient to make advisable its use in preference to the other.

Another and much larger problem in which the War Department is interested is that of the relative values of various fuels for domestic heating purposes. On account of the widely separated locations of the army posts a great variety of fuels are used and, since the intention is to furnish for officers and men a fair allowance of fuel for the conditions at their respective stations, information as to the relative values of these fuels under the conditions of use is of importance. The department therefore requested that the bureau conduct tests in several heating boilers, for which they supplied a large number of samples of coal, wood, and oil. The results obtained will furnish information of interest and value not only to the War Department but also to householders throughout the country.

The extent of the interest of the householder in this question of the selection and use of fuel for domestic heating was well illustrated by the demand for a report published by the bureau during the past year entitled, "Saving fuel in heating a house."^a This report discussed briefly and generally the more important factors influencing the consumption of fuel in residence heating. The tests now being conducted will furnish the basis for other reports which will discuss much more fully the effect of some of these factors. For instance, the importance of employing proper methods of handling fires has been strikingly shown by the tests being conducted, and experience with a wide variety of fuels will furnish data on how different coals should be handled and will show what losses may occur from improper methods of operation.

Another report completed and issued during the year was one relating to tests of apparatus for indicating or recording the content of carbon dioxide contained in chimney gases. Equipment of this character has been purchased for many Government power plants and for a much larger number of commercial power plants, but there was little information available as to the accuracy of the instruments, the conditions affecting their accuracy, or the amount of attention necessary to keep them in running order. The report covers tests of seven different kinds of apparatus. A similar series of experiments is to be carried out with several types of apparatus for observing

^a Breckenridge, L. P., and Flagg, S. B., *Saving fuel in heating a house*: Tech. Paper 97, Bureau of Mines, 1915, 35 pp., 3 figs.

judging, or recording the density of smoke emitted from power-plant chimneys. Both of these general kinds of equipment may be classed as apparatus to enable the engineers in charge of fuel-burning plants to supervise more closely the work of the operating force and thereby reduce unnecessary losses of fuel.

INSPECTION OF GOVERNMENT FUEL PURCHASES.

The importance of supervising more carefully the Government's coal purchases was early recognized, as coal was being purchased according to reputation and trade name rather than on the basis of its quality as measured by contained moisture, volatile matter, and ash, and by heating value and efficiency, as shown by use. Coal was purchased simply as coal, and no check was had on the character and quality supplied the consumer. Under the supervision of the Bureau of Mines uniform specifications have been developed. These prescribe in detail definite requirements for coal purchased by the Government. The plan of purchasing on the specification basis was first adopted by the Treasury Department in 1906, and this method of purchase has since been extended until the plan—variously modified in form but the same in principle—has been adopted by all the departments of the Government, and is applied to almost all contracts large enough to warrant sampling and analysis and heating-value tests. The value of the coal purchased annually by the Government under specifications prepared by the Bureau of Mines or under the advice of the bureau is now not less than \$7,800,000.

The work of fuel inspection includes the collection, analysis, and testing of samples representing coal purchased for the Government under specifications in order to ascertain whether the deliveries conform to the contract stipulations. In such a contract a bidder guarantees the quality of the coal he offers, and the quality guaranteed by the successful bidder becomes the standard of his contract. Analyses of samples of deliveries determine whether the coal delivered is of the guaranteed standard. If it is not, the price to be paid is proportionately decreased; if the coal is of higher quality, the price is proportionately increased.

A number of the States, many of the larger cities, and a large number of corporations and business concerns in different parts of the country have followed the general plans adopted by the Government for the purchase of its coal.

During the fiscal year 1916 a considerable number of samples of coal submitted by the States of New York, Indiana, Minnesota, Maine, Alabama, and Maryland were analyzed and tested. Authority for doing such work for State governments is given the bureau under its new organic act.

At the request of the Navy Department, the mining and shipping facilities of mines from which purchase of coal for that department is proposed are carefully examined by engineers of the bureau, and samples collected in each mine are analyzed at its laboratories. Requirements as to ash and volatile-matter contents are fixed, and these requirements must be met in all shipments. Occasionally samples are collected as the coal is being loaded on ships, and if it is not up to the standard, the contractor is promptly notified that his coal must comply strictly with the contract specifications. If he fails to ship coal of specified quality, his contract is annulled. Coal purchased under such specifications and used on naval vessels amounts to about 700,000 tons a year.

In order to determine the award of a particular contract, or to advise other bureaus and departments of the Government how a particular coal can be utilized most efficiently, or to ascertain what coal can be burned to best advantage in a particular type of furnace, or what changes in equipment will enable a particular Government building or plant to use efficiently the cheaper coals locally available, the bureau conducts steaming tests at its experiment station in Pittsburgh or cooperates with local engineers in conducting tests.

During the fiscal year 1916, 6,495 samples were analyzed. Each analysis was checked by duplicate determinations, and the laboratory sampling was checked by analyzing a duplicate sample. In addition, 172 samples of peat were tested in cooperation with the Minnesota State Geological Survey.

The success of the specification method depends largely on correct sampling of deliveries, for unless the samples are representative, the analyses and tests, however accurate, are worthless. Each year the local employees of Government power and heating plants are becoming more efficient in collecting samples and are thereby insuring the success of the specification method of purchase. However, experience and time will be required to perfect specifications and methods of sampling and analysis that may be used universally. Success now rests almost entirely on sampling. Improper sampling leads to controversies and to condemnation of the specification method. Mechanical devices for preparing and reducing samples are needed, for thereby the personal equation can be largely eliminated. The Bureau of Mines is developing such devices.

USE OF COKE FOR DOMESTIC PURPOSES.

In an investigation of the use of coke as a domestic fuel, information has been collected from coal-gas companies, from by-product coke companies, and from dealers handling domestic coke. Inspection trips have been made in the West and in the East. Coke manu-

facturers, fuel dealers, and householders have been interviewed concerning coke and its adaptability for household purposes. The information thus obtained has been prepared for publication.

Briefly, the general conclusion is that coke makes an excellent fuel for residence heating when properly handled, and in many sections its use for this purpose is increasing rapidly. Within the past five years there has been a remarkable development in the domestic utilization of coke and during the winter of 1915-16 the demand for coke for household use far surpassed all previous records and exceeded the supply.

PRELIMINARY LIGNITE INVESTIGATIONS.

One of the most important and far-reaching lines of investigation being carried on by the Bureau of Mines pertains to the proper systems of mining and the most efficient and economical methods of utilizing our fuel supplies and the various products derived from them.

In the west central and western States there are vast deposits of lignite, the extent and importance of which have been little appreciated. Economic methods for the development and utilization of these deposits and their important by-products have heretofore been given little scientific study.

For these reasons and in view of the fact that the Government controls great tracts of land underlain with lignite or immediately adjacent to lignite deposits, the Bureau of Mines has begun, in cooperation with the School of Mines of the University of North Dakota, in a small way, some preliminary investigations of certain phases of the lignite problem.

The meager funds that could be devoted to this work have not made possible any extended investigations, but the necessarily limited introductory work that has been done has demonstrated, beyond a doubt, the great economic importance of lignite and its products and has shown that adequate provision should be made for more extended research.

What has been accomplished in this preliminary experimental work leads to the belief that great improvements can be made in the methods of utilizing lignite as a fuel in the manufacture of briquets and in the production of cheap gas for power and other purposes. In addition to this, a large number of valuable by-products may be obtained by a variety of suitable methods of treating lignite.

Some of the phases of work during the past year are outlined below.

GAS FROM LIGNITE.

Tests as to the production of gas from lignite have been made by several methods under a variety of conditions of temperature, time,

and other modifying factors. The lignites have been found to yield much gas, its quality and character varying, of course, with conditions of production. There is little doubt, however, that this gas can be used for many purposes such as for power, heat, and light. The power tests indicate that it can be successfully employed to drive gas engines and that in such western lignite regions as are without water power, power can be produced at such a low figure as to exert a marked influence in the development of a variety of industries dependent upon a cheap and abundant power.

The gas thus used would, of course, be in the nature of a by-product. There can be obtained about 10,000 cubic feet of this gas from a ton of dry lignite, and in the unpurified state it has a heating value of 300 to 400 British thermal units per cubic foot. Under proper methods of treatment lignite can be used successfully also for producer gas.

LIGNITE BRIQUETS.

Because of the characteristic tendency of lignite to slack on exposure to air and the large amount of moisture it contains as mined, this fuel will not successfully stand long storage and can not be profitably shipped long distances from the mines.

After the lignite has been dried and the gases and other products removed there is left a residue that does not coke but has a high fuel value, and when properly briqueted produces an excellent free-burning fuel which will stand storage and transportation. The proportions of volatile matter and fixed carbon can be regulated by the degree to which carbonization is carried on. If a fuel with short flame, similar to anthracite, is desired, the carbonization is continued until nearly all of the gas and volatile matter has been removed, but if the briquets are to be used where a rather long flame is desired, the carbonization is carried only so far that sufficient gas and volatile matter for the desired length of flame will be left.

AMMONIUM SULPHATE FROM LIGNITE.

The preliminary work thus far carried on indicates that there can be obtained as a by-product from the carbonization of the lignite in the production of gas probably about 20 pounds of ammonium sulphate per ton of lignite. This material is valuable as a fertilizer and has other industrial uses.

LIGNITE TAR.

A considerable quantity of lignite tar is derived as a by-product from carbonization. This crude tar carries a variety of important products, which vary in character and amount with the temperature and time of carbonization. Among these crude-tar compounds may be mentioned a series of oils, a large proportion of which are among

the so-called "light oils," and a variety of tar acids, such as phenols and several others of value.

The preliminary work thus far indicates the probability of an unusually large yield of these tar acids from lignite, which is seemingly much larger than that obtained from the corresponding bituminous coke-oven tar, so that these by-products appear to be of much importance. The crude tar has an unusually oily, penetrating character and seems well adapted to immediate use as an admirable wood filler.

Although there is evidence that certain varieties of dyestuffs are obtainable from lignite and lignite tars, the limited investigations so far made have not made it possible to take up the study of lignite dyestuff products.

After the various oils and acids have been removed from the crude tar, considerable pitch of a superior quality is left which seems unusually well adapted for several important commercial uses, as well as for the briquetting of the residue.

LIGNITE OILS.

During the past year, in addition to a study of the problems and products already mentioned, a little preliminary research has been made of the derivation and character of oils obtainable from lignite. Although this work has had only a beginning, it is of promise in view of the quantity, variety, character, and commercial value of several of these oils.

Crude heavy oil is obtained in considerable quantity from the distillation of lignite tar. If the tar has been produced from the carbonization of lignite at a temperature of about 1,200° F., yields of heavy oil of 2 gallons and more per ton of residue have been obtained. This crude oil no doubt contains several grades, but it has not been possible to take up the further subdivision and purification of this oil. The flash point ranges from 105° F. for the lighter to 120° F. for the heavier products. This oil could seemingly be used for a variety of fuel purposes.

Some tests have also been made of the use of this crude heavy oil in connection with the concentration and flotation of certain ores, and under proper conditions it has given some gratifying results.

By the direct distillation of lignite at low temperature (about 700° F.) a totally different variety of rather light oil is obtained in considerable quantity. Yields amounting to 5 to 6 gallons per ton of dry lignite have been obtained, and it is quite possible that this yield can be materially increased.

It will be noted that the temperature of distillation is exceedingly low and the yield of light oil relatively large. The cost of produc-

tion, therefore, may be expected to be moderate. This light crude oil has a low flash point, and is seemingly a valuable fuel oil in the crude form, as it could doubtless be used with only partial refining for internal-combustion engines, but on further refining it yields other important grades of oil, among which might be grouped a crude benzol or benzol equivalent, which is highly volatile. Yields of 2 gallons and upwards have been obtained per ton of dry lignite. As this is easily obtained from the crude light oil of low-temperature distillation its cost of production should be relatively low.

On account of its highly volatile character and its freedom from heavier hydrocarbons, this oil would seem to be a valuable product for automobile and other similar types of gasoline engines. The most economic method of obtaining this product is worthy of thorough investigation, for it is easily obtained in considerable quantity, and the carbonized lignite left after distillation is an excellent fuel.

CONCLUSIONS.

Thus, the limited preliminary work thus far done on lignites and lignite products indicates that the possibilities in connection with the utilization of lignite and the development of its by-products and of associated industries are very encouraging and are worthy of thorough investigation and research, in view of the great extent of the lignite fields and the great importance of the lignite products, not only to the western areas themselves but to the whole country.

ELECTRICITY IN MINES.

During the past year the mining industry has shown an unusual interest in the bureau's work in connection with mine electrical equipment. The bureau has approved and developed more equipment than in any preceding year. During the fiscal year 1916 the bureau turned into the United States Treasury as fees from electrical tests the sum of \$2,189.82.

INVESTIGATION OF PORTABLE ELECTRIC MINE LAMPS.

Four complete lamp investigations were made during the fiscal year 1916, and approval was granted to each of the following lamps:

The Manlite lamp, Approval No. 11.*

The Concordia hand lamp, Approval No. 12.

The Wico lamp, Approval No. 14.

The Concordia cap lamp, Approval No. 15.

The General Electric Co. lamp, Approval No. 13.

The Hirsch Co. lamp, Approval No. 16.

* Approval rescinded for cause Apr. 23, 1917.

Approval No. 10, issued to the Edison Storage Battery Co. for its cap lamp, was extended twice during the year to cover improvements made by the manufacturer. Two brands of bulbs in addition to the brand originally approved for the Edison lamp were examined and approved.

The operators of coal mines are adopting with surprising and gratifying rapidity portable electric mine lamps approved by the bureau. It is probable that more than 100,000 approved electric lamps will be put into service in coal mines during the next 12 months.

INVESTIGATION OF EXPLOSION-PROOF MOTORS.

Two types of explosion-proof coal-cutting equipment were completely tested, and approvals Nos. 101 and 101-A were issued to the Goodman Manufacturing Co. to cover the 210 and 500 volt sizes of their type. The approval of the other type of machine tested was withheld pending certain mechanical and electrical modifications. In addition, approvals Nos. 100 and 100-A covering equipment of the Sullivan Machinery Co. were extended to cover improvements.

The Sullivan Machinery Co. developed, in cooperation with the bureau, an unusually efficient and safe plug connection for use in gaseous mines. This plug constitutes a unique device for breaking with perfect safety in the presence of gas comparatively large electric currents at voltages not in excess of 650 volts.

A new gallery for the testing of explosion-proof equipment was designed and erected. This gallery will greatly decrease the time required for making the tests of explosion-proof motors. There are now on file 18 applications for the testing of explosion-proof equipments, and one application for the test of a general-service motor, the development of which the bureau has been urging for several years.

RULES FOR INSTALLING AND USING ELECTRICAL EQUIPMENT IN COAL MINES.

During the year a set of proposed safety rules for the use of electricity in bituminous coal mines was proposed by engineers of the bureau and published as Technical Paper 138. About 15 conferences were held between the bureau's engineers and prominent mining engineers and coal-mine operators, and a comprehensive research was made of the carrying capacity of bare copper wires for mine use. Many changes suggested by mine operators and by mining engineers of the bureau were incorporated. The proposed rules have received favorable comments from some of the largest coal-mine operators, and have been reviewed and recommended for trial by the American Institute of Electrical Engineers.

EXPLOSION-PROOF STORAGE-BATTERY LOCOMOTIVES.

The interest that has been manifested by mine operators in the development of storage-battery locomotives for mines and the fact that the use of storage-battery locomotives will eliminate the trolley wire (the most dangerous piece of electrical equipment in mines) has led the bureau to undertake the preparation of specifications for explosion-proof storage-battery locomotives. Representatives from a large proportion of the manufacturers have conferred with the bureau and professed their interest in the bureau's purpose and their intention of cooperating. They suggested that the bureau prepare a set of specifications embodying the requirements that the bureau considers to be essential for an explosion-proof storage-battery locomotive. These specifications are completed, and have been submitted to the manufacturers of storage-battery locomotives and to several of the larger coal-mine operators, whose cooperation has been solicited and promised.

DEVELOPMENT OF A DURABLE MINE-LAMP CORD.

During the past year the bureau has developed a durable mine-lamp cord for the use of portable electric mine lamps. Up to the present time this part of such lamp equipment has been its weakest feature, and little success has seemed to attend the efforts of the lamp manufacturers to improve the cord. The bureau solicited the assistance of the manufacturers of electric wires and cables, and after the bureau had provided them with such information as it had at hand at that time regarding the best design to follow, the bureau's experts made tests of all of the different designs of lamp cords that the manufacturers submitted. The investigation is now complete, and 400 samples of 26 types of cord have been tested, involving a total of 47,000,000 slappings of the cords. The result of all this work has been the development of a cord 40 times as durable as the best cord tested prior to the undertaking of the research. This development means that the cord has been raised from the position of the weakest part of the lamp equipment to that of one of the most durable and reliable.

ELECTRICAL METHANE DETECTOR.

For several years the bureau has been developing an electrical methane detector. The results of the work have been gratifying and the detector promises to become a device that instantly will detect less than 0.25 per cent of gas. The sensitiveness and durability of the instrument are beyond question; some difficulty has been experienced in making the sensitive part of the instrument durable and reliable, but the results obtained recently have been encouraging.

INVESTIGATIONS OF MINE AND NATURAL GASES.

Important work was done during the fiscal year 1916 on a process for the absorption of gasoline from natural gas. This process consists of bringing so-called "dry" natural gases, which contain a very small percentage of gasoline vapor that can not be handled by the ordinary compression method, in contact with a petroleum distillate much heavier and of higher boiling points than gasoline, thereby causing the oil used to absorb gasoline from the natural gas. This absorption may take place in suitable towers or through a specially designed apparatus for absorbing vapors or gases in liquids. After the oil has absorbed the gasoline from the natural gas it is pumped to a steam still, where the gasoline is separated from the oil by fractional distillation. Then the oil is pumped back through heat exchanges and cooling coils to the absorber to receive another charge of gasoline. In other words, the oil acts simply as a carrier of gasoline from the absorber to the still and is used over and over again. The process is continuous, as the oil traverses a circuit and the natural gas is continually passing into and out of the absorbers. By the process millions of cubic feet of natural gas can be utilized for gasoline extraction that hitherto have been considered too lean, the estimated output of gasoline by the process being 100,000,000 gallons each year.

The Bureau of Mines did not initiate work on this process, for one of the large gas companies installed it in 1913, but the bureau is the first body to conduct elaborate experiments for the benefit of the public. These experiments demonstrated the fact that it is not necessary to perform the absorption at pressures above 30 pounds per square inch unless the pressure of the natural gas is high. As many millions of cubic feet of gas can be used at low pressure the bringing of this fact to the public is of great importance. A comprehensive report discussing in detail the many tests that were made is to be published soon.

Another achievement has been the development of a gas detector for mine use. This is considered to be the first practical, accurate, and simple gas detector for use in mines. A test requires less than two minutes, and an accuracy of 0.1 per cent can be obtained. The instrument is made of metal and weighs less than 2 pounds. It has been described in a preliminary article^a by one of the bureau's experts.

The first model, which is now in the hands of a company that is placing it on the market, is meant for use with the miners' cap lamp battery, many thousands of which are found in American mines. About 200 detectors are in actual use in mines. Another model to

^a Burrell, G. A., Description of a portable gas detector: Jour. Ind. and Eng. Chem., vol. 8, 1916, p. 865.

follow will contain small dry cells included in the body of the instrument, and perhaps a third model containing a very small storage cell so that the operation of the device will be independent of a source of electrical energy not contained in the instrument itself.

The development of the device has been somewhat slow because a great deal of time has been spent in making it explosion proof, and this object is believed to have been accomplished. The device has other uses besides detecting dangerous atmospheres in mines. For instance, it can be used in detecting different combustible gases like natural gas, artificial illuminating gas, hydrogen, acetylene, or other gases in air wherever they may accumulate and wherever it may be desirable to test for them.

The bureau's chemists have done further work on the measurement of natural gas at high pressures. Each year in the natural-gas industry many millions of cubic feet of gas are measured under pressures up to 300 pounds or greater, and the assumption has always been made that the gas follows the law for a perfect gas as regards the relation of pressure to volume. This simple law, called "Boyle's law," states that the volume of a gas times the pressure applied to the gas is constant at all pressures. However, this law does not always hold, and the neglect of proper correction for variations has resulted in the erroneous measurement of many millions of cubic feet of natural gas. The Bureau of Mines determined the deviation from Boyle's law for many samples of natural gas, and also for the constituents that comprise natural gas, in order to show the proper correction to be applied in natural-gas measurements. The results of this investigation will be found in Bureau of Mines Technical Paper 131.^a

NITRATION OF WATER-GAS TOLUENE.

An investigation of the nitration of water-gas toluene was undertaken in view of the rather prevalent impression that toluene obtained in the manufacture of water gas is unsuitable for the manufacture of trinitrotoluene, because of the presence in it of hydrocarbons of the aliphatic series whose removal can not be effected by the usual methods of purification. It was further anticipated that a similar difficulty would be encountered in efforts to utilize toluene prepared by processes involving the cracking of petroleum.

However, early in the course of the investigation it became apparent that aliphatic hydrocarbons, if present at all in any sample under investigation, were present in negligible amounts, as far as these affect nitration. Consequently, the investigation became a more general one, embracing the nitration of any good grade of toluene,

^a Burrell, G. A., and Robertson, I. W., The compressibility of natural gas at high pressure, 1916, 12 pp., 2 figs.

and it was continued with the purpose not only of working out the conditions under which the best yield of trinitrotoluene could be obtained from the sample of toluene in hand, but also of developing a method that might be capable of successful industrial application to the nitration of any toluene of approximately the same degree of purity. The need of such an investigation was emphasized by the incompleteness of published details with respect to methods already applied successfully in the industries. The data available in the literature on the nitration of toluene, although valuable and suggestive, did not afford a satisfactory basis for comparing the merits of different manufacturing processes with respect to quantity and quality of yield of trinitrotoluene.

Investigation of various conditions affecting the nitration of the toluene and the yield of trinitrotoluene have resulted in the development of a method of nitration by which a good yield of high-grade trinitrotoluene is obtained. The results of the investigation are embodied in Technical Paper 146^a of the bureau.

CONSTITUTION OF COAL.

Closely related to other investigations in progress in the bureau is the investigation of the chemical nature of the organic constituents of coal. Information derived from such an investigation of the nature of coal is of importance in its relation not only to pure science but especially to the determination of the factors that govern the coking property of coal, the spontaneous combustion of coal, and the inflammability of coal dust; to the correct interpretation of the changes that take place during the destructive distillation of coal; and to the intelligent and economical utilization of coal in the industries. Progress in such an investigation is necessarily difficult on account of the complexity of the coal substance and the great resistance that it offers to the ordinary physical and chemical methods of study.

On the chemical side, the method followed in the bureau has involved the separation of the constituent substances of coal as far as possible by means of extraction with organic solvents and the subsequent study of the separate portions thus obtained. A preliminary investigation along these lines is reported in Technical Paper 5^b of the bureau.

Previous to the resumption of a second investigation a few months ago, a large amount of coal from the Pittsburgh bed had been extracted with pyridine, the pyridine extract had been fractionated by

^a Hoffman, E. J., The nitration of toluene: Tech. Paper 146, Bureau of Mines, 1916, 30 pp.

^b Frazer, J. C. W., and Hoffman, E. J., The constituents of coal soluble in phenol: Tech. Paper 5, Bureau of Mines, 1912, 20 pp., 1 pl.

means of other organic solvents, and several of these fractions had been examined in some detail. The investigation has been continued during the latter part of the present fiscal year, and further progress has been made in the effort to identify some of the constituents of the coal previously separated by means of solvents.

WORK OF THE BUREAU'S ANALYTICAL LABORATORY.

At its Pittsburgh experiment station the bureau maintains a general analytical laboratory which is engaged principally in making the necessary analyses and tests that are constantly required in connection with the study of fuels and with the mine-accidents investigations that are being conducted at the station or in the field.

Naturally, the analysis and testing of coal and coke comprise the larger proportion of the work done at the Pittsburgh laboratory. The laboratory is equipped with the most improved apparatus for the complete and accurate analysis of coal, coke, lignite, and peat. During the year the bureau's equipment was increased by two new constant-temperature calorimeters, which are capable of giving the highest precision in determining the heating value of fuels. The work done in this laboratory during the year may be classified as follows:

ANALYSES OF MINE SAMPLES OF COAL.

In order to obtain adequate and comparable information on the value of the fuel resources of the country, the United States Geological Survey and State geological surveys, in cooperation with the bureau, have for a number of years been collecting mine samples of coal in various parts of the country. From time to time these analyses have been collected and tabulated in reference to their sources for publication in the bureau's reports. These publications form a library of ready reference to which busy operating engineers or purchasing agents of railroads, steamships, and large industrial concerns may turn and determine in advance the probable quality of coal available in any given locality.

The first of this series of publications, Bulletin 22,^a contains all analyses of mine and car samples of coal collected between July 1, 1904, and June 30, 1910.

The second, Bulletin 85,^b contains the analyses made between July 1, 1910, and June 30, 1913.

The third bulletin, covering the period from July 1, 1913, to June 30, 1916, is nearly ready for publication.

^a Lord, N. W., and others, Analyses of coals in the United States, with descriptions of mine and field samples collected between July 1, 1904, and June 30, 1910: Bull. 22, Bureau of Mines, 1912, 1,200 pp. (in two parts).

^b Fieldner, A. C., and others, Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913: Bull. 85, Bureau of Mines, 1914, 444 pp., 2 figs.

DETERMINATION OF COMBUSTIBLE MATTER IN STONE DUST.

In connection with the investigations on the limiting and prevention of dust explosions in coal mines by scattering stone dust in the mine passageways it became necessary to determine accurately the percentages of combustible carbon and hydrogen in clays, shale, limestone, and other materials that might be available for such use. Obviously, material that contains the least combustible matter is best adapted for the purpose mentioned. The usual methods of determining total hydrogen and carbon or ash do not give sufficiently accurate results, as they do not differentiate between combustible organic carbon and noncombustible organic carbon from the carbonates, nor do these ordinary methods distinguish between the organic hydrogen and the hydrogen in any combined water present in shales and clays. A simple method has therefore been devised whereby only the true combustible is determined.

DETERMINATION OF INCOMBUSTIBLE MATTER IN ROAD AND RIB DUSTS.

In applying stone dust for limiting explosions in mines it is necessary to determine quickly and on the spot the approximate percentage of noncombustible matter in the road and rib dusts in order to determine how much additional limestone or shale dust is required to prevent propagation of dust explosions.

For this purpose the Tafannel volumeter has been modified and combined with a convenient portable field equipment.

As shown in Plate XII, *C*, this outfit consists of the volumeter *a*, pipette *b*, balance *c*, sampling scoop *d*, sampling cloth *e*, alcohol can *f*, combined carrying case and table *g*, etc.

The determination of the incombustible matter is made as follows:

Twenty-five cubic centimeters of alcohol is measured into the volumeter flask with the pipette and is followed by 20 grams of the dust to be tested. The graduated tube is then inserted in the flask and the dust and alcohol thoroughly mixed by shaking; 25 c. c. more of alcohol is then added from the pipette through the stem of the volumeter, all adhering particles being carefully washed down. After 1 minute the scale reading of the meniscus is taken, and by reference to tables the percentage of incombustible is obtained.

DETERMINATION OF MOISTURE IN COMMERCIAL SHIPMENTS OF COKE.

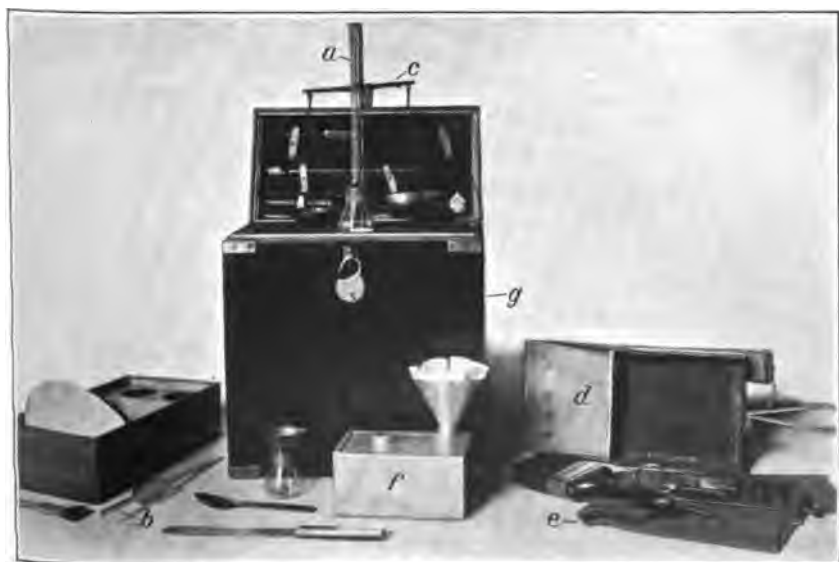
In cooperation with the committee on coke of the American Society for Testing Materials experiments were made to develop methods of determining moisture in shipments of coke. It was found that the moisture in coke could be determined by much simpler



A. ELECTRIC BRASS-MELTING FURNACE
DEVELOPED BY BUREAU OF MINES.



B. GAS WASTING FROM WELL BECAUSE
OF INEFFICIENT DRILLING EQUIPMENT.



C. VOLUMETER FOR DETERMINING PROPORTION OF INCOMBUSTIBLE MATTER IN MINE DUSTS.

methods than those required for coal, owing to the fact that the water in coke is all superficial moisture and in no way combined with the coke substance.

As a result of these experiments the bureau recommends that the moisture in coke be determined by drying the sample to constant weight at any temperature between 105° and 200° C. The sample should not be crushed to pieces smaller than 1 inch. No special oven is required. The foundryman may use a core oven to advantage, or a warm place in the boiler room. The principal precaution to be observed is to dry the sample as quickly as possible after it is taken and to keep it in a tight container while it is conveyed to the place for drying. The experiments have further shown that wet coke can not be crushed without losing moisture. The results of this investigation will be published in a technical paper of the bureau.

DETERMINATION OF NITROGEN IN COKE.

In connection with by-product coking investigations with special reference to the yields of ammonia under various conditions, it is frequently necessary to determine the nitrogen remaining in the coke. The usual methods of determining nitrogen in coal are exceedingly tedious when applied to coke; therefore experiments have been undertaken with a view to devising some shorter method to apply to coke determinations.

The action of various catalytic and oxidizing agents is being tried, and it is hoped so to modify the Kjeldahl method as to shorten the time of digestion considerably.

CHLORIDES, CARBONATES, AND ORGANIC SULPHUR IN AMERICAN COALS.

Certain coals, especially the English coals, contain chlorides. These chlorides seem to affect the brick ovens and clay retorts used in the manufacture of coke and illuminating gas. A series of analyses for chlorine and also for organic sulphur and carbonate carbon in coal has been made. In the coals tested thus far only traces of chlorine have been found. Organic sulphur, however, is a rather general constituent of American coals, in some of which it comprises half the total sulphur. The largest proportion of carbon in the form of carbonates, has generally been found in coals from the interior field, Illinois, Indiana, Missouri, and western Kentucky.

FUSIBILITY AND CLINKERING OF COAL ASH.

The high rates of combustion that are used in modern boiler furnaces have greatly increased fuel-bed temperatures and consequent trouble from clinker formation, especially in certain types of me-

chanical stokers in which the fresh coal is pushed up into the hot fuel bed. There is, therefore, a demand for a simple, duplicatable laboratory test whereby the clinkering tendencies of various coals may be graded numerically.

The fusibility of the ash, as indicated by the temperature at which it softens or deforms when molded in the form of Seger cones, has been generally regarded as offering some indication of the clinkering properties of a coal. It is, however, now well known that the results of this test are greatly affected by the shape of the test piece, the rate of heating, and especially by the nature of the atmosphere—whether oxidizing or reducing—in which the ash is heated. Experiments were made showing the effect of the various fuel-bed gases, such as, oxygen, carbon dioxide, carbon monoxide, hydrogen, and water vapor, on the fusion of ash. The highest temperature of fusion was found in an oxidizing atmosphere, and the lowest temperature of fusion in an atmosphere that was partly reducing, such as to reduce the iron to the ferrous state in a nonmagnetic slag. This type of slag is generally found in fuel-bed clinkers, and therefore indicates the condition under which the laboratory test should be made.

Two furnaces and methods of operation have been devised to make these softening-temperature tests so as to obtain nonmagnetic slags, as follows: (1) Fusion in an atmosphere of hydrogen and water vapor in a molybdenum electric furnace, and (2) fusion in a gas furnace operating with an insufficient air supply the unburned CO and hydrocarbons from the gas forming the reducing atmosphere.

This latter form of furnace gives in general somewhat higher results (up to 100° C. higher) than the electric furnace with an atmosphere of hydrogen and water vapor. However, the gas furnace is so much cheaper to build and operate that, in spite of the greater variability in the results, it will probably find more general commercial use than the electrically heated furnace. In operating the gas furnace a reducing atmosphere must be maintained by using the least air that will support combustion enough to attain the desired temperatures. It is necessary merely to suspend the fused cone from a thread in the field of a strong electromagnet to determine whether the cone has any appreciable magnetism. If the cone shows more than a trace of magnetism, the atmosphere is not reducing enough and the test should be repeated with a larger excess of gas.

The results of this investigation are being compiled for publication in a bulletin.

**DETERMINATION OF FERROUS, FERRIC, AND METALLIC IRON IN
SLAGS AND CLINKERS.**

In order to determine the state of oxidation of the iron in clinkers and in cones fused in various atmospheres, it was necessary to determine the amounts of iron present in the three possible forms, namely, metallic iron, ferrous oxide, and ferric oxide. The three following methods were studied: (1) The copper sulphate method; (2) the mercuric chloride method, and (3) the bromammonium-acetate method. The latter was found most accurate, although for certain purposes where only the percentage of metallic iron is desired the first two methods are applicable. As the textbooks on quantitative analysis make no reference to methods for determining metallic iron, this work should be of considerable value to anyone desiring to make such determinations. The details of the work will be outlined in a technical paper of the Bureau of Mines.

MINERAL-TECHNOLOGY INVESTIGATIONS.

INVESTIGATION OF RADIUM AND OTHER RARE METALS.

Until three or four years ago radium was more or less a scientific curiosity, the amount in existence being small, and scattered among a reasonably large number of people. The result was that no one person had what would now be considered a reasonable amount of the material. Some men, such as Sir William Ramsey and Sir Ernest Rutherford, possessed something like 200 milligrams of radium metal, all of their intensely interesting and valuable scientific work having been done with this amount. The reason for the dearth of radium was that the ore supply was limited, and, moreover, methods of manufacture were expensive. With the advent of the successful use of radium in cancer therapy, the demand for radium increased, and about this time carnotite ore began to be shipped from southwestern Colorado and eastern Utah to Europe where the radium was extracted. The Bureau of Mines became interested in the country's uranium deposits, and started an investigation, the results of which were published in Bulletin 70* of the bureau. The report showed that the largest deposits of uranium-bearing ore in the world were situated in southwestern Colorado and eastern Utah, although this fact was not generally known in either of these States. In fact, it was not until Bulletin 70 appeared that the real value of these deposits was publicly appreciated.

RADIUM EXTRACTION BY BUREAU OF MINES.

After it had been shown that a considerable ore supply was available, it became advisable also to show that the treatment of radium-bearing ore and the refining of the material obtained were feasible in this country. This necessity culminated in the cooperative arrangement between the Bureau of Mines and the National Radium Institute, whereby the Government was able to experiment on the treatment of carnotite ore, and incidentally to retain in this country a supply of radium for two of our important hospitals. This work was begun in 1914-15 and has been continued in 1916. The result has been that more than eight grams of radium has been obtained at a cost of less than \$40,000 per gram. This radium is not for sale but is being used in the General Memorial Hospital in New York and in the Howard A. Kelley Hospital in Baltimore.

* Moore, R. B., and Kithil, K. L., A preliminary report on uranium, radium, and vanadium: Bull. 70, Bureau of Mines, 1913, 101 pp., 4 pls., 2 figs.

This amount is more than seven times as much as the total amount owned and in use in this country before the work began. The scale on which the radium extraction has been carried on is indicated by the fact that the production in a single month has been twice as large as the amount of radium in the possession of Rutherford or Ramsey, and until a few years ago the amounts they had at their disposal were considered relatively large. Not only has this large amount of radium been conserved to this country for the use of cancer treatment and for scientific purposes, but all of the details necessary for the treatment of carnotite ore have been worked out and published for the benefit of anyone interested. Little had previously been published concerning the method of treating radium-bearing ores and the refining of the salts obtained, but Bulletin 104* of the Bureau of Mines gives information not only as to the treatment of the ore, but also as to methods of analyses and details concerning the refining of the crude radium salts obtained. After the work had been started it was found that the generally known methods of analyses for radium were not applicable for commercial purposes, and one of the earliest and most important parts of the work involved an investigation of methods of analyses that would be applicable to the existing conditions.

The radium plant was operated until January, 1917. At that time the contract between the National Radium Institute and the Bureau of Mines was completed. A certain proportion of the radium produced has been retained by the Bureau of Mines under its agreement with the institute. Part of this will be used for scientific work, and part for medical work in connection with cancer. There are a considerable number of problems in connection with radium that have not only a scientific value but also a commercial bearing. The material retained by the bureau will give an opportunity for the study of a number of these problems. In addition, a considerable amount of radioactive lead, ionium, and actinium concentrates have been obtained, and methods for further concentrating and refining these will be studied, and the material obtained will be valuable for scientific work.

TREATMENT OF CARNOTITE AND OTHER RADIUM-BEARING ORES.

The actual work of mining carnotite ores, which was started in 1914 in cooperation with the National Radium Institute, was continued in 1915 under the direction of the bureau's engineers. A bulletin on the work will soon be issued. There has been mined in all about 1,000 tons of shipping ore, averaging over 2 per cent U_3O_8 , and in

* Parsons, C. L., and others, Extraction and recovery of radium, uranium, and vanadium from carnotite: Bull. 104, Bureau of Mines, 1915, 124 pp., 14 pls., 9 figs.

connection with the mining of the commercial ore, about 2,000 tons additional of low-grade milling ore heretofore wasted, containing an average of 0.85 per cent U_3O_8 , has been produced and concentrated. A concentrating plant was built at the mines at Long Park, Paradox Valley, southwestern Colorado, and has been operated on a commercial scale, turning out about 300 tons of concentrates averaging 3 per cent U_3O_8 . Waste ores of this grade have heretofore been left on the dump, but it has now been definitely proved that such ores can be made into a commercial product by dry concentration. An enormous waste may be thus eliminated, as ores of this grade have not heretofore been profitably marketed either as ore or as concentrates. The forthcoming bulletin will describe the mining and concentrating methods in detail and give costs.

The mines department of the National Radium Institute has furthermore purchased, under the cooperative agreement with the Bureau of Mines, about 300 tons of shipping ore, which gave a limited market to the individual miner at a time when the European war had eliminated all other purchasers. The concentration method as demonstrated by the bureau should be of great benefit to the carnotite industry, as it will enable the miner to make the mining of this rare ore more profitable. He necessarily must mine a large amount of low-grade ore at the same time that shipping ore is produced. By concentration this waste product may be utilized, the output of radioactive ore being thus largely increased.

The bureau has also obtained for investigation over 100 tons of low-grade pitchblende ore, together with over 7 tons of high-grade pitchblende. Most of this ore was mined at Quartz Hill, Gilpin County, near Central City, Colo., and has been kept in storage in Denver for some time. The poorer ore was so low in its uranium and radium content that it could not be chemically treated without further concentration. The Colorado School of Mines at Golden offered the use of its mill to the Bureau of Mines, and experiments for the concentration of this ore have been conducted there with the apparatus available. Most of this ore was exceedingly low grade, consisting chiefly of iron pyrite, silica, and less than 0.5 per cent U_3O_8 in the form of pitchblende. The concentration of this ore has been successfully accomplished and a considerable saving has been made, although it is hardly feasible to treat commercially such low-grade ore. On the other hand, it is believed that more careful hand sorting of this low-grade ore would yield a better product for which a ready market may be found.

INVESTIGATION OF RARE METALS AND THEIR ALLOYS.

A considerable amount of work was also done during the fiscal year 1916 on the treatment of molybdenum-bearing ores. Molybdenum is

a metal that until recently has been mainly used in the chemical industries, but its use as an alloy for special types of steel demands attention also. The metallurgy of molybdenum has been little understood and the methods used have been expensive. Two types of ore are commonly found in the West, namely, molybdenite or molybdenum sulphide, and wulfenite or lead molybdate. The former frequently carries impurities, mainly copper, which must be separated from the molybdenum in order that the latter can be sold. Wulfenite contains not only lead, but sometimes gold, vanadium, and other metals, and therefore its metallurgy is somewhat complex, as any really successful method should obtain all of the valuable substances present.

General problems in connection with uranium, vanadium, tungsten, molybdenum, and other rare metals will be investigated, particularly as regards the ore supply, methods of concentration, and ore treatment. It has already been pointed out that a small quantity of copper in molybdenum ore is sufficient to prevent the sale of such ore, and the same thing applies to certain impurities in tungsten ores. The question of cheap and efficient methods for the elimination of such impurities would put on the market ore that at present can not be sold to advantage.

Many of the rare metals form alloys, both among themselves and with the commoner metals, and the alloys have a great commercial value. For example, tungsten when added to steel gives to the steel certain qualities that cause the product to be in great demand for the manufacture of cutting tools for lathes, etc. Recently uranium has been successfully substituted for a large part of the tungsten in tool steel to the seeming improvement of the product and it is reported that uranium steel is showing remarkable endurance when used for the liners of cannon. Some alloys of the rare metals containing no iron at all have shown interesting properties, and much research in regard to these and other alloys should be undertaken.

The bureau contemplates further investigations of the mining and concentration of all the rarer metals, such as tungsten, molybdenum, antimony and quicksilver, in order to utilize ores of a grade that has been discarded in many of the mining districts of the West. The European war, which has cut off many of our supplies heretofore imported, has aroused much interest in the conservation of our minor metal resources. The development and application of processes for saving the low-grade rarer minerals offer special inducements, and as all investigations carried on are made especially with the view of preventing waste and utilizing low-grade products, this work should be of great benefit to the mining industry.

Many mineral specimens have been sent to the Bureau of Mines for identification, and although under the law the bureau has no authority

to make analyses, great assistance has been rendered to prospectors and miners by simple identification of the material. Although most of the specimens sent in are of no value, it can be stated that a number of deposits have been opened as a result of the proper identification of some of these minerals, and the results have been gratifying.

STUDIES OF NONFERROUS ALLOYS.

The nonferrous-alloy investigations of the bureau are being conducted by one alloy chemist and one assistant alloy chemist at Cornell University, Ithaca, N. Y., under a cooperative agreement with the university by which the facilities of the chemical laboratories, and in particular of the extremely well-equipped electric-furnace laboratory, are available for the bureau's work.

The problem under investigation concerns mainly the metal losses in the nonferrous-alloy industry, particularly in the brass industry. This work has been in progress nearly four years, starting with a study of existing methods of brass melting to ascertain the metal losses, fuel consumption, crucible life, etc., in the various types of fuel-fired furnaces. The results of this study are given in Bureau of Mines Bulletin 73.*

From the point of view of the theorist, electric melting of brass should offer marked advantages over existing methods, mainly in the reduction of losses of zinc from alloys high in zinc content, such as yellow brass, manganese bronze, and German silver, in lessened danger of oxidation in the melting of such alloys as the bronzes and red brasses, and in the possibility of replacing the present method of melting in crucibles of comparatively small capacity, with attendant high labor cost, as used in brass rolling mills, by a method capable of using much larger melting units.

The correctness of the theory of the decreased melting losses in electric furnaces of suitable types has been established by a long series of experiments in various types of furnaces.

DEVELOPMENT OF ELECTRIC BRASS-MELTING FURNACE.

The problem then became one of finding an electric furnace capable of reducing the metal losses and giving the other advantages of electric melting with such a high consumption of electric power as to make its commercial operation show an over-all saving in melting cost. Almost any electric furnace will melt brass. Many types will reduce zinc losses, though some will not. But to find a furnace that combines all the required factors of metal saving, power

* Gillett, H. W., Brass-furnace practice in the United States: Bull. 73, Bureau of Mines, 1914, 298 pp., 2 pls., 23 figs.

efficiency, low upkeep cost, reliability, and general applicability to the needs of the brass industry has been no light task.

In the search for such a furnace many types have been built and operated by the bureau on an experimental scale.

Several commercial firms have made tests of electric brass furnaces on a commercial scale, and almost every one of these firms has co-operated fully with the bureau, allowing representatives of the bureau to attend these tests. Neither the laboratory experiments of the bureau nor the trials by private parties showed much promise of commercial success until the fiscal year 1916. During that period, however, two different types of electric furnaces tried by private parties have shown promise. One of these types is fitted only for use with material low in zinc, and its margin of saving over present methods is somewhat problematical under normal conditions. At the present price of crucibles it may profitably replace crucible furnaces under favorable conditions.

The other type of furnace is applicable to alloys high in zinc, and although so far tried only in small sizes offers great promise for certain phases of the brass industry. It has its distinct limitations, however.

The bureau has built and operated on a laboratory scale a third distinct type of electric furnace which has shown marked promise. This furnace promises to meet conditions for which the other two types are not well suited.

The laboratory furnace developed by the bureau is shown in Plate XII, A. The furnace rests on a track set on brick piers. Below the furnace track is another track on which a buggy runs to carry ingot molds. The furnace is rocked back and forth by the projecting handles shown. Adjustable electrodes between which an electric arc is formed are introduced at the ends of the furnace along its axis. The opening shown closed by the clamp in Plate XII, A, constitutes a combined charging door and pouring spout.

Arrangements have been made with a large central electric power station for the building and cooperative testing of a furnace of the type worked out by the bureau. The bureau is keeping in close touch with the progress of the other types of commercial furnaces and hopes soon to be able to publish a report that will bring out the possibilities and limitations of electric brass melting. It seems almost a certainty that in a few years electric brass melting will have a strong foothold in the brass industry and that the electric furnace will prove a means of notably decreasing the Nation's loss of metal in brass melting, which in normal times is about \$3,000,000 a year and in the past year of large production and high metal prices has probably been nearer \$10,000,000.

The American Institute of Metals is officially cooperating with the bureau in its work. Great interest has been shown in the electric brass furnace work by brass founders, rolling-mill men, and electric-power companies.

PYROMETER DESIGNED BY BUREAU.

In the work on the electric brass furnace it became necessary to measure satisfactorily the temperature of the molten metal. For such service a pyrometer that will combine accuracy, long life, and speed of reading is essential. No pyrometer on the market is fully satisfactory for this purpose, but in the bureau's work one was devised that has given good service for laboratory use.

Figure 6 shows the details of the bureau's pyrometer. The protecting tube consists of a nickel tube which is tipped with molybdenum and encased in a tube made of silfrax (graphite coated with SiC), as shown in the figure. The thermocouple consists of a platinum element and a platinum-rhodium element carried in Marquardt porcelain insulating tubing, as shown. The graphite and molybdenum only come in contact with

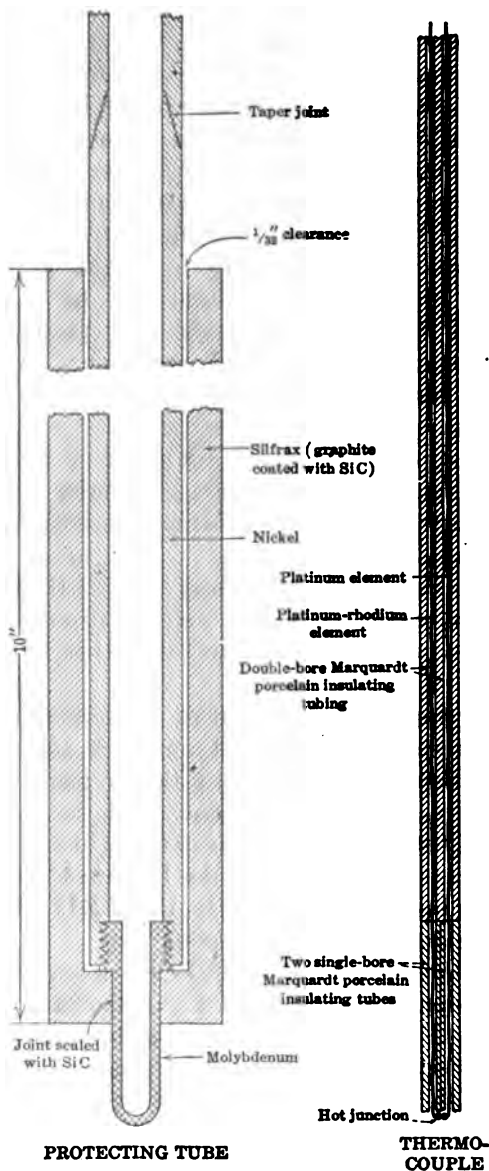


FIGURE 6.—Laboratory pyrometer developed by Bureau of Mines.

the molten metal, and neither is attacked by it. The nickel tube resists oxidation. The small molybdenum tip heats up quickly, the lag of the device averaging 50 seconds in metal at 1200° C. with the tip

cold at the start. Other nickel tubes are threaded into the lower one to give a tube 3 or 4 feet long, and the outer end is fitted with any suitable handle.

Temperature control is of vital interest to the brass founder and to the rolling-mill operator, as on the temperature rests not only the quality of the goods produced, but also to a greater or less degree, the metal losses, fuel consumption, etc. The bureau has, therefore, taken up the problem of finding a pyrometer for use in molten brass and bronze which will not only serve the requirements of the laboratory but also the needs of the foundry and the rolling mill. This problem seems now to be in a fair way to satisfactory solution.

MELTING OF ALUMINUM CHIPS.

Another problem dealing with metal losses in nonferrous alloys has been a study of methods of melting aluminum chips such as are produced in the machining of the aluminum alloys used in motor-car construction.

These fine chips are hard to remelt without excessive loss, a 40 per cent loss being not uncommon. This loss costs the Nation in normal times about \$200,000, and in the past year, on account of the extraordinarily high price of aluminum, has probably been nearer \$600,000.

Several refiners have been able to get a recovery of 85 to 90 per cent of the metal content of the chips, but knowledge of the necessary details of the methods has been confined to a very few. The bureau has studied the various methods in detail, with the cooperation of several prominent refiners, and the results of the work, published in Bulletin 108,^a will, it is believed, inform anyone having to deal with this problem how to procure high recoveries and avoid excessive loss.

An interesting point in this work was the establishing by a series of experiments that the reason commonly accepted for the difficulty of melting aluminum chips without high losses—oxidation of the metal within the furnace—is probably far from being the main reason. The main cause of trouble seems to be the lack of ready coalescence of the metallic globules, which means that the problem is one of colloid chemistry applied to metals.

The cooperation of Prof. W. D. Bancroft of Cornell, an authority on colloid chemistry, was of great value in the work on this problem.

PRODUCTION OF FERRO-URANIUM.

Another problem, allotted to the Ithaca laboratory because of its facilities for electric-furnace work, is the study of the production of

^a Gillett, H. W., and James, G. M., Melting aluminum chips: Bull. 108, Bureau of Mines. In press.

ferro-uranium. Uranium oxide having been produced as a by-product in the radium cooperative work of the bureau and the National Radium Institute, the best method for the utilization of this uranium was sought.

It is understood that uranium steel has recently found use abroad in gun liners, although chemical and metallurgical literature is singularly barren of information on the subject. The results of the investigations undertaken by the Bureau of Mines are to be published as a technical paper.*

CERAMIC RESEARCH.

COMMERCIAL REFINING OF GEORGIA KAOLIN.

During the past fiscal year the bureau has devoted its attention to only one ceramic problem, the commercial refining and use of the kaolins of Georgia by the control of the colloidal material in them, and the utilization of the prepared clay in the manufacture of vitreous china and wall tile.

Notwithstanding the fact that this country possesses enormous deposits of secondary kaolins, especially in Georgia and in South Carolina, this material has not been considered available for the manufacture of white ware, and our ceramic industries have been dependent on imported china clay. For the five years previous to 1915 the annual imports of English china clay from Europe were in excess of 250,000 tons, at an average cost of \$6 per ton.

The question, "Why can not American kaolins be substituted for English china clay?" has been frequently asked and more often since the beginning of the present European war, which has threatened to cut off the supply of clays imported from Europe. The answer to this question has invariably been that the domestic kaolins are *not* so pure and uniform in composition as the imported clays.

The demand for white-burning American clays is of such economic importance that it was considered desirable to determine whether some of our vast deposits of impure white clays could not be refined sufficiently to permit their being substituted for the foreign materials.

The mechanical methods of refinement used hitherto have not yielded products of the desired degree of purity and uniformity. It was decided, therefore, to attack the problem with the methods of colloid chemistry.

It is well known that the addition of small amounts of alkalis to clay-and-water systems has a deflocculating action; that is, it effects a disintegration of the clay grains, or increases the dispersion of the system. The amount of clay held in suspension is increased

* Gillett, H. W., and Mack, E. L., Preparation of ferro-uranium: Tech. Paper 177, Bureau of Mines. In press.

and the viscosity of the system is correspondingly decreased. Acids and salts have an opposite effect, causing an increase in the viscosity of the system and a flocculation and rapid settling of the clay particles. The fact that alkalis when added to clay-and-water systems cause deflocculation may be applied to the purification of Georgia clays. The impurities in the kaolins are coated with clay, so that it is impossible to settle out the impurities without a great loss of clay. When the clay-and-water system is dispersed, as by the addition of an alkali, the impurities are free to settle out.

Investigations of methods of refining secondary kaolins were conducted in the Washington laboratory of the bureau in 1913-14. It was found that by thoroughly deflocculating kaolin from the Dry Branch district of Georgia, with a small-measured quantity of caustic soda, the impurities could be settled out with great ease. The product showed considerable improvement in color when made into hard fire-glazed porcelain bodies.

In order to determine whether some practical method of refining the secondary kaolins of Georgia on a commercial scale could be devised, the Bureau of Mines arranged a cooperative agreement with the Georgia Kaolin Co., of Dry Branch, Ga., covering investigational work, which was begun in March, 1915. The effect of additions of different amounts of various alkalis, acids, and salts on Georgia kaolin suspensions was studied, the change in the viscosity of the system being taken as a measure of the effect of the electrolyte added. The electrolytes used included hydrochloric, sulphuric, and acetic acids, alum, caustic soda, sodium carbonate, and sodium silicate.

FACTORS STUDIED.

The investigation included a study of the factors affecting both the initial and the minimum viscosity, such as predrying and blunging, under the conditions of a commercial plant. It was found that caustic soda caused the greatest drop in viscosity of any of the reagents used. Of the coagulating agents tested, hydrochloric acid proved to have the strongest action, being slightly greater in its effect than sulphuric acid.

PROCEDURE IN INVESTIGATION.

In order to insure the successful operation of the refining process developed, suitable equipment for the refining of 50 tons of crude kaolin was installed at an operating mine in the heart of the Georgia clay field. The refining process was carried on under the same conditions that would have prevailed if the ordinary clay miner had been using the process. The refined white product obtained was sent

to a number of the most prominent users of such clay, and was found to possess properties so different from those of the crude kaolin that an additional investigation was necessary to insure the successful utilization of the refined product.

With this object in view an experienced ceramic engineer was sent to two pottery works where the refined kaolin had been shipped; mixtures were made that yielded products of a quality equal or superior to those made from imported material. Two private commercial plants are now being equipped for the refining of Georgia kaolins by this process.

Whether the work already done by the Bureau of Mines will suffice to demonstrate how this process may be applied commercially to all the clays in the South Carolina and Georgia district is a matter of speculation. Doubtless some of the crude clays in that district will not respond to this treatment. In fact, some are known, but the bureau would hardly be justified in undertaking a special investigation to cover their commercial purification until the extent of their occurrence is established. However, preliminary work has been conducted so that as soon as such proof is furnished, the work of practical adaptation of the process to these special clays may be effected with the least possible delay.

STONE-QUARRY INVESTIGATIONS.

PURPOSE OF THE INVESTIGATIONS.

In 1914 a cooperative agreement was entered into between the United States Geological Survey, the Bureau of Standards, and the Bureau of Mines for a study of the stone-quarrying industry of the country. The work undertaken by the Bureau of Mines had for its chief objects the promotion of safety and efficiency, and the elimination of waste in the industry, as well as a study of the technologic methods used and the problems involved.

IMPORTANCE OF THE QUARRY INDUSTRY.

There are approximately 3,000 quarries in active operation in the United States, employing in all about 100,000 men. Quarrying is, therefore, an industry of considerable magnitude. One can scarcely overestimate the importance of quarry products in the Nation's life and work.

Quarries supply, in the form of building stone, the most permanent and attractive of all structural materials. When this fact is appreciated more fully by architects and builders, natural stone will replace much of the cheap and temporary material of which many buildings are now constructed, and as a result buildings will have greater

beauty and permanence, and the ultimate cost will be less than where cheaper materials are employed. For interior decoration natural stone excels all other materials in color effect and in capability of being shaped into artistic designs. For monuments or sculpture nothing can take its place. For the production of Portland cement, which is now indispensable in many lines of industry, millions of tons of rock are quarried annually. Street and road construction, for which millions of dollars are spent every year, depend to a great extent upon quarry products, both for crushed stone and for cement. The production of lime, ground limestone, and gypsum for soil amendment, and limestone for blast-furnace flux, is continually increasing in magnitude.

It is evident, therefore, that in attempting to safeguard the lives and health of quarry workers and to promote the highest efficiency and economy in the quarry industry, the Bureau of Mines has initiated a movement that directly benefits a great group of industries and a vast multitude of individuals and that indirectly benefits many more.

During the fiscal year two reports on stone quarrying were published by the bureau.

ACCIDENT PREVENTION URGED.

The first publication, "Safety in Stone Quarrying,"* appeared in September, 1915. Its purpose was to point out to the stoneworkers of the country the chief sources of danger, particularly in the marble-quarrying industry, and to emphasize ways and means of materially reducing the number of casualties. The death rate due to accidents among quarrymen for the year 1913 was 1.72 per 1,000, which is higher than in most European countries for which records are available.

In studying the causes of accidents it was found that they could be grouped in three main classes. In the first class are the many accidents from faulty equipment. In this class may be placed boiler explosions, falls of derricks, and accidents due to insufficiently protected rotating machinery, electric wires, and quarry stairs and ladders. A second cause of accidents is faulty methods, such as improper handling, loading, and firing of explosives, and dangerous methods of handling rock or overburden. The third group comprises accidents that result from carelessness on the part of employers and employees. The report has received favorable comment from quarry operators, and the bureau has received requests for many extra copies to be distributed among quarry superintendents.

* Bowles, Oliver. Safety in stone quarrying: Tech. Paper 111, Bureau of Mines, 1915, 48 pp., 5 pls., 4 figs.

INVESTIGATION OF MARBLE QUARRYING.

A second publication, "The Technology of Marble Quarrying,"^a was issued in March, 1916. At the time this was the only publication in the English language that took up a specific branch of the quarrying industry and dealt with it exhaustively.

IMPROVED METHODS RECOMMENDED.

An important feature of this report is the emphasis placed upon the influence that rock structures should have upon quarry methods. This consideration has been generally overlooked by quarry operators, with consequent heavy but preventable losses. For example, in quarries having inclined open-bed planes that make the maintenance of level floors impossible, it has been customary to construct supports in order that channeling machines may be operated on level tracks. Some of the more progressive companies have recently operated channeling machines on inclined tracks placed flat on the slanting quarry floor, without costly supports. A "balance car" overcomes the effect of gravity. This improved method, as shown by careful records, has resulted in a 50 per cent increase in efficiency in certain quarries of this type.

Other structures that influence quarry methods are "joints" or cracks which commonly intersect marble deposits in two directions, approximately at right angles. If marble blocks are intersected by such seams, their value is greatly impaired, and consequently an effort is always made to quarry sound blocks. Many quarrymen have taken no pains to determine the compass direction of the major joints, and consequently the channel cuts commonly cross them at oblique angles, and thus many blocks are unnecessarily marred by seams. The bureau has emphasized the fact that in the processes of channeling or wedging out blocks the cuts should be made parallel with and at right angles to the joints. The advantage of such a method is shown in figure 7. It may be readily seen that when channeling is properly conducted, as indicated at B in the figure, most of the joints can be eliminated entirely and blocks of sound marble obtained.

REDUCTION AND UTILIZATION OF WASTE.

The problem of waste has received special notice. The presence of vast piles of waste marble at many quarries indicate the two important phases of the problem: First, the desirability of reducing waste by introducing improved quarry methods, and, second, the need of utilizing accumulations of waste material. The bureau has shown that the proportion of waste may be reduced by intelligent prospecting and by separating blocks in accordance with rock structures.

^a Bowles, Oliver, The technology of marble quarrying: Bull. 106, Bureau of Mines, 1916, 174 pp., 12 pls., 88 figs.

Blocks containing imperfections may be employed for riprap, road building, lime burning, soil amendment, furnace flux, rubble, and various other uses.

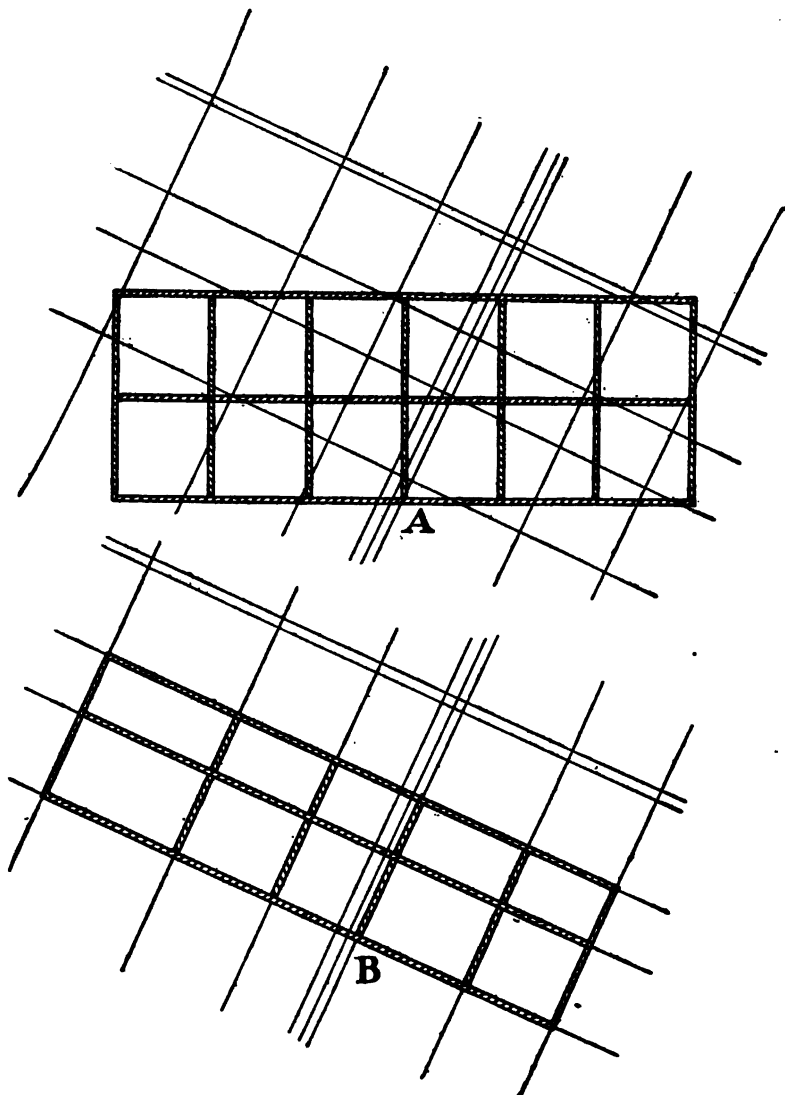


FIGURE 7.—The economy of channeling in accordance with joint systems. A, Plan of channeling without reference to the direction or spacing of the joints, involving great waste of marble blocks; B, plan of channeling in which channel cuts are made parallel with and, as far as possible, coincident with joints, avoiding waste of blocks.

REDUCTION OF LOSSES DUE TO "STRAIN BREAKS."

An important feature of the report is a detailed record of a successful method of reducing the excessive loss due to "strain breaks"

in marble quarries. The rock in many quarries is under compression, and as the compressive strain is relieved locally in quarrying, the partly severed blocks break into irregular masses, the fractures often being accompanied by loud reports. A method of avoiding the excessive waste due to the production of these irregular fragments is described. It is advised that rows of deep, closely spaced, vertical drill holes be projected in a line across the quarry in such a manner that the rock may expand and partly close the drill holes, thus giving relief from strain without the destructive fracturing. One Tennessee marble company has already tried this method, and has thereby greatly reduced the proportion of waste marble, and thus effected a saving of several thousand dollars. It is believed that several valuable quarries that have during recent years been abandoned on account of excessive strain breaks could be reopened and worked profitably if the methods proposed in Bulletin 106 were adopted.

IMPORTANCE OF KEEPING COST RECORDS.

The report puts stress on the fact that waste reduction and efficiency of operation can not be achieved if definite cost data are not kept. The point is emphasized that the majority of quarrymen fail to keep definite cost records, and as a result they are unable to properly judge the efficiency of methods and machines. The introduction of more modern methods of quarrying is therefore frequently discouraged by the inability of the operator to test their efficiency. To facilitate the keeping of technical records a complete system of marble-quarry accounting is given.

SANDSTONE QUARRIES EXAMINED.

During the summer of 1915 a study of sandstone quarrying was begun. About 55 sandstone quarries were visited by the bureau's quarry technologist, and investigations were made of all phases of the industry. A special study was made of the physical properties and imperfections of sandstone and their effect on quarry operations. Methods of quarrying bluestone and gneiss were also observed.

It is worthy of note that these investigations disclosed a condition of excessive waste in many sandstone regions, the proportion of waste in some instances reaching 75 per cent of the gross production. The gravity of this condition is intensified by the limited number of uses for which waste sandstone may be employed. The imperative need and the means of so improving quarry methods that the proportion of waste may be greatly reduced are emphasized. A study of the

cause and prevention of accidents was conducted in conjunction with the efficiency and waste investigations. The result of the studies and observations made are embodied in a report.*

OTHER RESEARCHES BEGUN.

During the latter part of the fiscal year similar investigations were begun at the fluxing-limestone quarries in the vicinity of Buffalo, N. Y., and the cement-rock quarries of eastern Pennsylvania and northern New Jersey. Observations at the small number of quarries already visited indicate that the problems involved in the quarrying of these types of stone demand careful study, and that a detailed discussion of quarry methods, particularly those relating to blasting, would be of material benefit to the industry.

* Bowles, Oliver, Sandstone quarrying in the United States: Bull. 124, Bureau of Mines, 1917, 148 pp.

PETROLEUM INVESTIGATIONS.

IMPORTANCE OF PETROLEUM PRODUCTS.

The average person fails to realize the importance of petroleum and its products in the ordinary routine of the world's work, yet every man, woman, and child in America is dependent in some measure upon petroleum and natural gas. There is scarcely a phase of our national life in which we do not find petroleum products used—they drive our battleships, deliver our merchandise, pull our trains, heal our wounds, color our garments, smelt our ores, carry our mails, cook our meals, and increase our knowledge of the outdoor world. They illuminate the magazine we read and the book we study; they carry us home from the office, and make chewing gum for the children.

Every industry makes some use of petroleum or its products, and to list all of the uses would not be feasible here. It is sufficient to say that petroleum is essential to the commercial development of the country, and that if our supply were cut off to-day, our industrial progress would be brought almost to a standstill. It has been said that when petroleum is gone, electricity will take its place, but, although electricity will furnish power for our industries and lights for our homes, it will not lubricate even the machinery needed for its own generation.

The Navy is dependent on the petroleum industry not only for lubricating oils but to an increasing extent for fuel. Battleships that use oil as a fuel can carry larger armaments, and have a greater range of action than those that use coal. Petroleum is thus vital to our national defense and perhaps to our national existence.

LIMITATIONS OF SUPPLY OF PETROLEUM.

What shall we do when our supplies become depleted? Although this country has large known reserves of petroleum, these supplies are limited. At our present rate of consumption our estimated supplies are sufficient to meet our present needs for a comparatively short period, conservatively estimated to be from 25 to 30 years, taking no account of the increasing demand for petroleum and its products. This estimate not only includes the oil fields already known and developed, but makes liberal allowances for undiscovered fields in prospective oil territories. It should not be thought that our petroleum supply at the end of that period will be cut off abruptly,

for the wells will continue to produce through a declining output for many years. On the other hand, the shortage of petroleum is beginning to be felt now. A good measure of the accuracy of these estimates, which were made more than a year ago by the Department of the Interior, is the result of the search for oil the past year, Owing to the demand for crude oil, more territory was prospected and more wells were drilled in search of petroleum in the Mid-Continent field than in any previous year. During the year 1916 there were approximately 15,000 wells drilled in that field, as compared with 6,000 wells drilled in 1915, yet the production to-day will not equal that of two years ago, in 1915, nor have any large new oil fields been developed.

What effort have we made to conserve this supply and to utilize it to its greatest advantage? We have made little effort until very recently to do these things. We have been wasteful, careless, and recklessly ignorant. We have abandoned oil fields while a large part of the oil was still in the ground. We have allowed tremendous quantities of gas to waste in the air. We have let water into the oil sands, ruining areas that should have produced hundreds of thousands of barrels of oil. We lacked the knowledge to properly produce one needed product without overproducing products for which we have little need. We have used the most valuable parts of the oil for purposes to which the cheapest should have been devoted. For many years the gasoline fractions were practically a waste product during our quest for kerosene; with the development of the internal-combustion engine the kerosene is now almost a waste product in our strenuous efforts to increase the yield of lighter distillates.

This country is producing about two-thirds of the world's output of crude petroleum and has produced approximately 2,750,000,000 barrels since the drilling of the first oil well by Col. Drake in 1859. Our future supply from both proved and prospective oil fields, based on geological possibilities, is estimated to be approximately 7,402,000,000 barrels, which will last us only about 25 years at our present rate of consumption. We are exporting 20 per cent of our production, using approximately 25 per cent as fuel under boilers, and of the remainder probably one-fourth is wastefully utilized. Thus 70 per cent in all is being used in a manner that must be considered anything but conservative for so valuable and so scarce a product.

As an example of wasteful utilization, a large proportion of our artificial gas is made from petroleum, notwithstanding the fact that coal has been and is used for this purpose, and would be used altogether except for the reason that the gas manufacturer is able to buy petroleum cheaper. At the present rate of production it is estimated that our coal supply is adequate for many centuries. Clearly we

should not use our petroleum for fuel under boilers or for gas manufacture, or in any way to compete with coal, when the oil supply is so limited.

GROWING CONSUMPTION OF GASOLINE.

Gasoline is indispensable to our present industrial progress. At the present time there is a greater demand for gasoline than for kerosene, fuel oil, or any of the other petroleum products. There are now over 3,500,000 automobiles in use in the United States. The manufacturers are turning out approximately 1,500,000 new machines this year, and expect to increase the number each succeeding year. The average consumption for each automobile is more than 10 barrels of gasoline a year, and it is estimated that this country will need more than 2,000,000,000 gallons of gasoline for the year 1917. Where is this gasoline coming from?

METHODS OF PRODUCING GASOLINE.

The situation can be partly met by converting some of the less valuable products of crude oil into gasoline. The Bureau of Mines early realized the importance of this conversion and during the year 1915 one of its chemical engineers, Dr. Walter F. Rittman, discovered a "cracking" process by which gasoline, as well as benzene, toluene, and other desirable products can be made from kerosene and other crude oil distillates. Other cracking processes have been invented but they have been privately owned and not readily available to the small refiner. The Bureau of Mines hopes to make the Rittman process available to every user.

It is estimated that during 1915, 2,000,000 barrels of gasoline was made by cracking processes, some refineries obtaining as high as 35 per cent of gasoline from some oils. During the year 1916 it is estimated that 5,000,000 barrels of gasoline will be manufactured by cracking lower grade products. This output is all the more striking when it is considered that these 5,000,000 barrels will be made from oils that in the past did not enter into the making of gasoline.

The Bureau of Mines has for a number of years realized the importance of taking steps to introduce methods by which wells may be drilled more economically and with less waste of oil and gas to increase the amount of oil that may be extracted from an ordinary formation, and above all to conduct investigations that will bring about a greater efficiency in the utilization of this precious commodity. To carry on work along these lines the petroleum division of the Bureau of Mines was organized, and up to July 1, 1916, had spent about \$100,000 in this work. The manner in which this money has been expended and the resulting benefits to the public form an interesting study for anyone interested in the petroleum industry.

ORGANIZATION AND WORK OF THE PETROLEUM DIVISION.

The petroleum division of the Bureau of Mines is divided into four sections, as follows: (1) Cooperative and administrative work, (2) production technology, (3) engineering technology, and (4) chemical technology. The cooperative and administrative section supervises the work of the division, conducts its correspondence, and carries on work in cooperation with other governmental agencies.

The section of production technology conducts investigations with a view to increasing the proportion of petroleum and gas recovered from the earth and carries on studies regarding the exclusion of water from the oil sands and the prevention of other underground wastes. The section also investigates current oil-field practices throughout the country with a view to the more general adoption of the most efficient and economical methods of producing the oil and caring for it. Men with long practical experience in drilling and caring for oil properties are held ready to advise oil operators who desire expert opinion as to the best methods to be used in solving any difficult problem.

The section of engineering technology conducts investigations as to the design, construction, and operation of pipe lines, storage tanks, earthen reservoirs, and similar problems connected with the engineering side of the petroleum industry. A special study has been made of steel storage tanks and of the loss of oil in storage, including both losses by evaporation and losses by fire. Also, the section has studied in detail the manufacture of gasoline from natural gas.

The section of chemical technology studies methods and practices for refining and treating petroleum and its products and is the section in which the Rittman process was developed. This section carries on research work in connection with the refining and testing of petroleum, and is working on the study of the processes for deriving oil from shales with a view to determining the future economic possibility of the oil-shale industry. This is a large and important field of work, and it is expected in the future to materially assist in introducing new practices in refining that will, to a certain extent, aid in the efficient utilization of our petroleum.

COOPERATIVE AND ADMINISTRATIVE WORK.

The purposes of the petroleum division have been three: First to place before the public the significance of the tremendous waste of oil and gas and if possible to cooperate with State legislative bodies and Federal bureaus to reduce and to prevent excessive wastes of oil and gas; second, to carry on investigations whereby a greater percentage of the oil may be extracted from the sands; and, third, to

place in commercial use various methods that will insure a greater utilization of this country's petroleum resources.

Millions of cubic feet of natural gas and millions of gallons of oil have been wasted for days and days in a careless and reckless exploitation of our oil and gas fields. The productive districts in this country have now been fairly well defined, and in spite of the constantly increasing demand for these unrivaled fuels the production must within a comparatively short time begin to decline. Hence, it is of the highest importance to the Nation that the remaining supplies of these fuels be protected and the tremendous waste in their production be prevented, and it is this end that the administrative and cooperative section holds in view.

Many of the States have already recognized the need of preventing waste and have enacted laws for that purpose. Unfortunately, however, such legislation has in many instances proved ineffective, because it did not have the confidence of oil operators and because it was difficult to enforce. The Bureau of Mines is in a position to bring together producers and marketers of petroleum and natural gas so that their cooperation may be obtained and measures to lessen wastes may be made effective.

EXAMPLES OF AID TO STATES.

A few months ago when the people of Oklahoma realized the tremendous waste of natural gas and oil in the prolific Cushing field, it became necessary for the State legislature to prepare and enact into law a bill making it compulsory for the oil operators to conserve these resources. The assistance of the Bureau of Mines was requested and was promptly given. The law is now in force and is spoken of as one of the best conservation laws in the United States.

The bureau stands ready at any time to furnish any possible similar assistance to any organization or State for the purpose of conserving petroleum and natural gas.

Another interesting example of assistance given by the Bureau of Mines also concerns the State of Oklahoma. The commercial club of a small but thriving city, realizing that much of the surrounding district's prosperity depended upon the conservation of its oil and gas, requested that the bureau recommend to them a practical man whom they could employ for the purpose of enforcing the regulations of that State in connection with the conservation of its petroleum resources. The committee asked that this man be allowed to work under the supervision of the Bureau of Mines. A man was finally appointed, his salary and expenses being paid from moneys subscribed by the commercial club and by representatives of the various industries in the community, and under the bureau's direction many millions of cubic feet of natural gas have been conserved and saved.

for the future, while in the near-by districts, where no such close scrutiny was given the oil and gas operations, all the gas has either been blown into the air in an effort to find oil or has been dissipated in the porous formations underground. The Bureau of Mines desires to see more of this sort of work done, as it not only results in the conservation of oil and gas but insures the future prosperity of many of the industries upon which such cities are dependent.

INTERDEPARTMENTAL COOPERATION.

Cooperative work has been carried on with such Federal departments and bureaus as the Navy Department, the Department of Justice, the General Land Office, and the Indian Office. The bureau's petroleum division seeks to act as the technical advisor of the Government and of the people in all matters relating to the technology of production, engineering, and refining of petroleum and natural gas.

PRODUCTION TECHNOLOGY.

The section of production technology of the bureau's petroleum division studies the drilling of oil and gas wells and the production of oil and gas.

PROBLEMS IN DRILLING GAS AND OIL WELLS.

The drilling of oil and gas wells is not a simple matter, and only a small percentage of the people of this country realize that it costs thousands of dollars to drill a hole over half a mile deep for the purpose of discovering oil and gas. Many oil wells are nearly a mile in depth. Oil and gas exist ordinarily in porous formations at varying depths below the surface, and if it were possible to drill wells for oil and gas in the same way that wells are drilled for water, not so many problems would be encountered. But the holes must be drilled through caving formations and through formations containing great quantities of water or gas under high pressure. Heavy pipe ranging in diameter from 4½ to 20 inches must be used to prevent the caving of the formations and to exclude water from the drill hole.

There are two general methods in use in the United States for drilling oil or gas wells—the standard or cable-tool method, in which a percussion drill is used, and the rotary system, in which, as the name implies, the drilling is done by rotating a string of pipe on the end of which is a bit that cuts through the formations in the same manner that a drill bit cuts through a piece of metal.

With the standard method the first drilling is done with a large tool called a "bit," which is raised by powerful machinery and allowed to drop, grinding up the rock. As soon as the hole reaches a depth at which caving begins, a heavy "string" of casing is

screwed together and set on the bottom of the hole, and another bit of a smaller size, which will go inside the casing, is lowered into the well, and drilling is resumed in the smaller sized hole. When found necessary the bit is removed and a long string of casing, smaller in diameter than the first casing, is set on the new bottom of the hole and operations continued with the smaller bit. This is done until the oil sands are encountered, the strings of casing resembling a great telescope.

Many difficulties are connected with drilling such wells, and the bureau's engineers are continually studying the problems of operators in different fields of the United States. These engineers are sent from field to field to advise with operators when requested to do so. In this way the Bureau of Mines disseminates much information of value to the oil operators.

NEED OF TECHNICAL CONTROL IN WELL DRILLING.

As outlined above, the drilling of oil and gas wells is a difficult engineering operation; it requires the expenditure of much money and labor, and demands high engineering skill. However, all drillers and many operators are not familiar with the most efficient methods of drilling through gas-bearing formations and often do not adequately protect oil and gas measures from contamination by water or from underground dissipation. For this reason and because of the great volumes of oil and gas being wasted, the bureau undertook to introduce into Oklahoma new methods of drilling that would result in a conservation of oil and gas. Some specific instances of wells at which large volumes of gas were wasted follow.

EXAMPLES OF GAS WASTE.

In 1914 one of the first wells drilled in the north Cushing field wasted an average of 14,000,000 cubic feet of gas each day for 67 days, or a total of 938,000,000 cubic feet. A little later the same well struck another sand and wasted about 40,000,000 cubic feet of gas each day for seven days, or a total of 280,000,000 cubic feet. From these two sands this well wasted 1,218,000,000 cubic feet of gas, which, if it had been sold at the average price of gas consumed in the United States at that time would have brought \$182,700, and even if the producer had sold the gas at the customary field price he would have obtained from its sale over \$25,000, or more than enough to drill a well. This amount of gas wasted is equivalent to about 60,000 tons of coal, or about 250,000 barrels of oil, and is sufficient in volume to furnish over 12,000 families with gas for one year. Had this well been the only one to represent waste, it would not have been

so regrettable, but as a matter of fact the gas wasted by this well was only a small proportion of the total amount wasted in the Cushing field.

For instance, a well near by, drilled at nearly the same time, in less than a month wasted 1,655,000,000 cubic feet of gas, equivalent to about 80,000 tons of coal. Still another well within two months wasted 600,000,000 cubic feet of gas, and another in 32 days wasted 1,000,696,000 cubic feet of gas. The total combined waste from these four wells was over 5,000,000,000 cubic feet, equivalent to over 250,000 tons of coal, or enough to supply over 50,000 families for one year. If this gas had been sold at the average price of gas consumed in this country it would have brought the seller more than three-fourths of a million dollars.

During the year 1913 it is estimated that in this one field in Oklahoma an average of not less than 300,000,000 cubic feet of gas was wasted daily, or more than 100,000,000,000 cubic feet of this ideal fuel was allowed to waste during the year. This is equivalent to about 5,500,000 tons of coal, and would have met the wants of nearly 1,000,000 families for 1 year. If the gas had been sold at the rate of 15 cents per 1,000 cubic feet, the sellers would have realized over \$15,000,000, and even if the producers had obtained only 3 cents per 1,000 for the gas, which is the prevailing field price, they would have realized over \$3,000,000 from its sale. Not only was the gas allowed to waste, but such tremendous volumes of this inflammable material hung over the oil fields that automobiles were not allowed to enter, and in many cases disastrous fires were started, resulting in the loss of life and property.

All this gas was wasted in order to produce about 30,000 barrels of oil daily; in other words, at the prevailing price paid by domestic consumers for such fuel, gas worth about \$75,000 a day was needlessly wasted to obtain a daily oil production valued at less than \$25,000.

EFFECTIVE METHODS DEMONSTRATED BY BUREAU OF MINES.

The Bureau of Mines sent engineers to this field to reduce the waste by teaching an efficient drilling method. In a few months it was proved beyond doubt that wells drilled through these tremendously productive sands need not waste so much of the gas. The method demonstrated not only conserved the gas and kept it underground where it could not be wasted, but rendered the field safe for workmen and for visitors, and also greatly reduced the time of drilling wells, for previously when an operator struck a high-pressure gas sand he had to allow the gas to needlessly waste into the air until the

pressure had declined enough to allow him to continue drilling. An example of the waste of gas is shown in Plate XII, *B*. This well was wasting 50,000,000 cubic feet of gas daily and had a closed-in pressure of nearly 800 pounds.

The method of drilling advocated by the Bureau of Mines is called the "mud-laden fluid" method, and is described in detail in Bulletin 134 of the bureau. In short, this method may be outlined as follows:

METHOD OF USING MUD-LADEN FLUID IN DRILLING.

The term "mud-laden fluid" is applied to a mixture of clayey material that will remain suspended in water for a considerable time and is free from sand, limestone cuttings, or similar materials.

ACTION OF MUD FLUID ON POROUS FORMATIONS.

The action of mud-laden fluid in a sand or other porous formation can be likened to the action of muddy water going through a filter. In any filter that has been used for some time it will be found that most of the sediment from the water has been deposited on the surface of the filter, but some of it has entered the filter, the proportion diminishing with the distance penetrated. The distance to which mud from the fluid in the well will penetrate a porous formation depends partly on the combined pressure produced by the column of fluid and the pump, and partly on the consistence of the fluid and the porosity of the formation. At first the fluid will enter the formation, but finally the mud will clog the pores and no more water will go through. Ordinarily, if a thick fluid is used on the sands encountered in the well, it will not penetrate to any great distance even under high pressure, but if the fluid is too thin it may not clog the pores readily and will act more like clear water, which may enter a sand indefinitely. Occasionally a very coarse sand, a fissured formation, or a porous limestone is found into which even thick fluid may penetrate for some distance.

When a well has been treated with mud fluid the contents of each formation is confined to its original stratum, so that there can be no movement of oil, water, or gas either from the sands into the well, from the well into the sands, or from one sand into another. Thus waste and intermingling are prevented, corrosive waters can not reach and attack the casing, and the strata are entirely sealed off from each other as they were before the well was drilled.

CONSISTENCE OF FLUID TO BE USED.

The consistence of the fluid should be varied according to the conditions for which it is to be employed. Most frequently mixtures

with a specific gravity of 1.05 to 1.15 are used in drilling. When the fluid is not used to drill in, thicker fluid is often employed, which has the advantages of greater weight and of clogging the pores of the sand more readily. Experience soon enables the operator to judge the consistence of fluid required for practical uses.

The operator who is unfamiliar with the use of mud-laden fluid is likely to use it too thin. This has been the cause of much trouble in Oklahoma. Such fluid acts almost like clear water. It will not clog the pores of the sand readily and hence will be forced into them for considerable distances, and in some instances near-by wells have been affected. It is also likely to cause caving and is injurious to the sand, or it may not have sufficient weight to overcome high-pressure gas. The only limit on the thickness of the fluid that may be employed is whether it can be handled by the pumps, for it must be a fluid and not a pasty clay.

MUD FLUID CONTRASTED WITH OTHER METHODS.

The mud fluid is used in Oklahoma at present in drilling wells and in protecting formations behind casings and plugging wells. The careful use of mud fluid has proved of inestimable value in drilling oil wells; it prevents dissipation of gas underground as well as escape at the surface by confining the water, gas, and oil to the strata in which they originally occur.

Mud fluid may be used in wells that have been improperly cased to obviate the bad results which ordinarily follow this practice. An example of improper casing is shown in figure 8. If mud fluid had been placed behind certain of the casings in the wells shown in the figure, waste of gas would have been prevented.

Mud fluid has proved to be of value in plugging wells, and if it were used more constantly there would be no such waste as that shown in Plate XIII, A, which shows a gas well that was improperly plugged. After the well had been abandoned the gas pressure forced a leak, and gas-laden water flowed from the hole.

It is estimated that in the north Cushing field only about 10 per cent of the recoverable gas was actually utilized for any purpose, whereas if the operators had used the mud-laden fluid system of drilling it would have been possible to utilize 80 per cent of this gas. Since the introduction by the Bureau of Mines of this process even larger percentages of gas have been utilized.

An attempt has been made to impress upon the oil operators of Oklahoma and of some of the other States the fact that formations containing only a small amount of oil should be carefully conserved and protected because the day of the big oil gusher will soon be past

and oil operators will then be drilling new wells to sands now comparatively small in production, and moreover they will be making money from such wells.

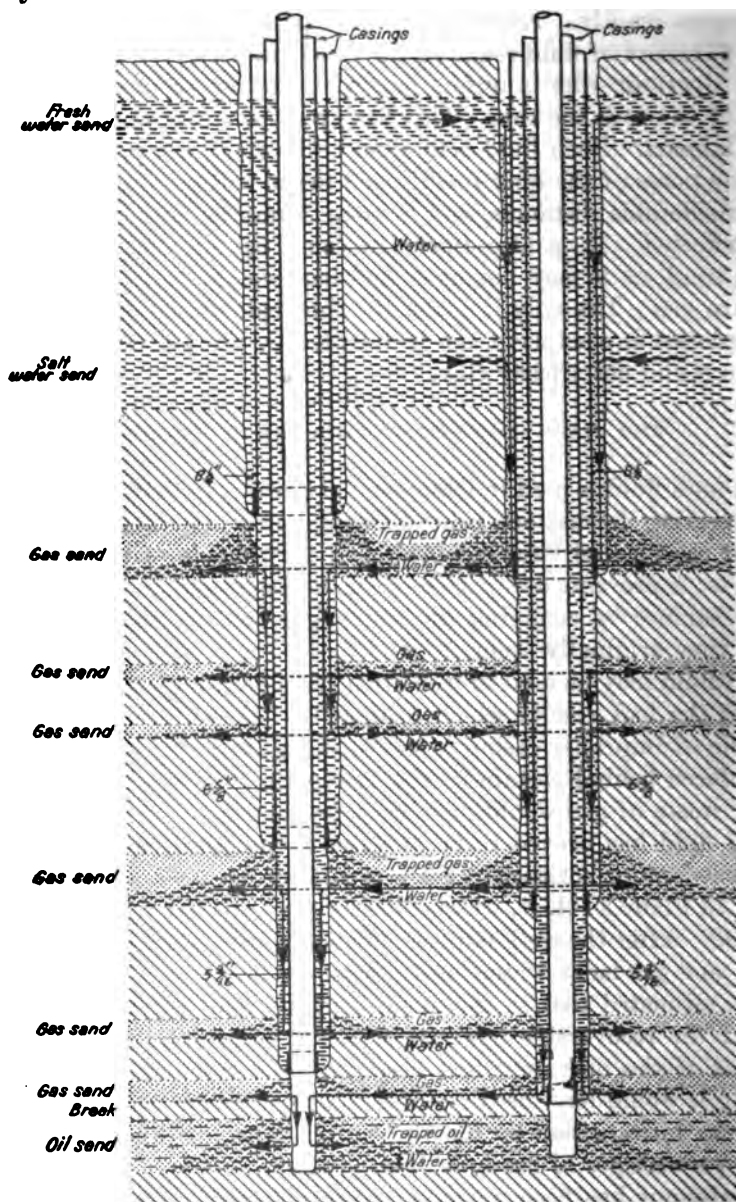


FIGURE 8.—Diagram showing how gas may be wasted by improper casing. The gas is in sufficient volume and under enough pressure to overcome the water; it works up, from the sands of highest pressure, behind the casing and back and forth through the other formations. Much of the gas is lost, and its pressure is greatly reduced. Some of the gas may reach the surface and escape in the air.



A. WASTE FROM IMPROPERLY PLUGGED GAS WELL.



B. TEXAS OIL FIELD, SHOWING WELLS THAT COST THOUSANDS OF DOLLARS EACH.



C. REMAINS OF OIL-STORAGE TANK DESTROYED BY FIRE.



COST OF OIL WELLS AND NEED OF EFFICIENT DRILLING METHODS.

In the drilling of oil wells, with the expenditure of vast sums of money, many problems arise. Plate XIII, *B*, shows the Sour Lake oil field in Texas. Each well shown cost thousands of dollars, and required many weeks of labor in drilling. After oil is discovered many more problems are encountered by the producers in extracting the oil from the ground. At some wells as soon as the drill penetrates the oil sand, oil and gas in tremendous volumes are expelled from the hole, and if the producer is lucky enough to have storage tanks, he may make money easily while the well is flowing.

Oil properties are unlike mining properties, where a certain amount of ore is mined each day and that amount can be increased by putting more men to work. Very often an oil well "comes in" at a high rate of production which lasts for a number of months, gradually diminishing in daily output, however, as the underground pressure in the oil sand is released. As a result the well makes a constantly diminishing daily production, and oftentimes after a few months it becomes necessary to place tubing in the well and pump the oil from the sands. Some wells produce for many years, but often one-half of a well's total production is obtained the first year, and it requires all the rest of the years of the well's life for the operator to obtain the remaining half of the oil.

It has been estimated that oil operators are obtaining only a part of the total oil stored by nature under their lands. Estimates of the total amount extracted range from 10 to 70 per cent, 90 to 30 per cent being left in the ground. It is estimated that this country in the future will produce over 7,000,000,000 barrels. If the Bureau of Mines is able to cause the adoption of practices whereby this production is increased by 10 per cent it will cause an increase in the ultimate production of 700,000,000 barrels or a sufficient amount to meet the present demand for about two years. Placing the same price per barrel on this crude as was obtained for the crude oil throughout the United States for 1915, there would be a saving to the country of \$450,000,000. Even an increased ultimate production of 1 per cent would yield a total net saving of \$45,000,000, a sum far in excess of any amount which will be spent by the Government for investigations of petroleum and natural gas.

EFFORTS TO INCREASE OIL PRODUCTION.

The engineers of the bureau are making extensive investigations with a view to recommending the most feasible methods of increasing the extraction of oil by the use of compressed air or the use of a vacuum. The principle of one of the most promising of these proc-

esses is the forcing of compressed air into the porous oil sand, thus forcing the oil in every direction from the point of entry of the air. The oil is then pumped from the wells near by. In some cases partial vacuum is put on the sand by the use of suction pumps. These pumps draw the oil and gas toward the well to which the pumps are attached.

Water is the oil man's greatest enemy. The filtration of water into the oil sands means either a total loss of the oil or a comparatively short flow. This has been true with hardly an exception for every oil field in the world. In many oil fields divided among a number of operators there are great differences in the methods used to get the oil, and occasionally an operator who has had very little experience and does not realize the harm he is doing does not carefully exclude from the deep oil sands the water in the shallow sands. The water not only ruins his well but rapidly spreads from well to well, decreasing the amount of oil recovered, "cuts" the oil, and spoils the field.

The Bureau of Mines has already issued a report* on the cementing of oil wells, which means the exclusion of water from the oil by the use of cement. This practice is followed almost universally in the California fields and in some of the Texas fields, and the bureau hopes by conducting extensive work to induce operators in other fields to adopt similar methods. As our petroleum resources are limited, we can not afford to allow great quantities to be ruined because some operator has been neglectful in the methods he has used in finishing his well. The bureau is to issue a report on the methods used for excluding water from oil sands as practiced in all the fields of this country, which will be written with a view to recommending the best practice to be followed under varying conditions in different fields.

CORROSION OF WELL CASINGS.

The corrosive effect of underground water on casings is another difficulty that oil men must combat. Casing in oil wells is used primarily to exclude water and loose material, and as some underground waters contain acids, casings in such wells are rapidly corroded and water runs through them into the oil sands. The casings may be protected from the corrosive effect of water by the use of mud-laden fluid, but in case operators desire to use some other method to obviate this difficulty, the bureau hopes to be able to suggest alternatives.

* Arnold, Ralph, and Garfias, V. R., The cementing process of excluding water from oil wells, as practiced in California: Tech. Paper 82, Bureau of Mines, 1913, 12 pp., 1 fig.

ENGINEERING TECHNOLOGY.**STUDY OF EVAPORATION AND FIRE LOSSES.**

The section of engineering technology of the petroleum division is studying evaporation losses and fire losses in connection with the storage of oil with a view to reducing these losses by suggesting improvements in the design and protection of tanks. The magnitude of evaporation losses may be realized by considering that out of 50,000 barrels stored in one tank in Oklahoma between 2,000 and 2,500 barrels was lost by evaporation in five months. There are probably between 3,000 and 4,000 large oil storage tanks in Oklahoma to-day, and although they do not all now contain oil, and it is not reasonable to suppose that such tremendous losses occur from every tank, the evaporation loss is undoubtedly appalling. In addition to this loss, many tanks are lost by fire caused by lightning during certain seasons of the year, and it is a common occurrence in the oil fields during a thunder shower to see a great tank containing thousands of barrels of oil rapidly burning. Plate XIII, *C*, shows one of these tanks after such a fire. Not only is the oil lost, but the tank itself has no value.

STUDY OF METHODS OF TRANSPORTING OIL.

Transportation of oil is being studied, including the influence of temperature and viscosity on the capacity of pipe lines, the influence of pressure and soil conditions on the life of pipe lines, and the proper power equipment for pipe lines. Pipe lines are expensive; they cost between \$5,000 and \$10,000 a mile, depending upon the conditions under which they are constructed and upon the topography of the country that they traverse. When it is remembered that pipe lines have been laid from Oklahoma to the Atlantic coast and to the Gulf coast, and that practically all the Pennsylvania and West Virginia oil fields are connected by pipe lines to the great refinery centers on the Atlantic coast, one realizes what tremendous engineering problems are involved in the construction of these lines and what vast sums of money are invested in them.

CHARACTER AND UTILIZATION OF "WET" GAS.

When oil is produced it is almost always accompanied by more or less gas which ordinarily comes from the same sand. The gas usually contains some of the lighter portions of the oil itself. Such gas is known as "wet" gas, and if it is produced with the oil it is called "casing-head" gas. A few years ago one could go through almost any oil field in this country and see great volumes of this gas

escaping from the oil tanks. No one could see any way in which it could be utilized, although at some wells it was trapped and used for fuel under boilers. It was soon learned, however, that by compressing this wet gas and cooling the compressed product gasoline of high gravity was precipitated. A few years ago gasoline was not so expensive as it is now, and inasmuch as the so-called "casing-head" gasoline was more or less dangerous on account of its volatility, not many people cared to handle it. However, after the automobile came into its own and gasoline became so expensive, the use of casing-head gasoline began to be more and more common.

PROCESSES FOR MAKING GASOLINE FROM CASING-HEAD GAS.

Casing-head gasoline is made by two different processes, the compression process and the absorption process. In the compression process the wet gas is compressed by large machines to a pressure between 200 and 300 pounds to the square inch. The compressed product is then cooled and the "wet" parts of the gas condensed. The high-gravity product is then mixed with low-gravity distillates and kerosene, which can not be utilized by themselves in automobile engines. However, the addition of the light-gravity gasoline gives the resulting mixture a volatility sufficient to permit ready combustion in automobile motors.

The absorption process of making gasoline may be used for casing-head gas or for gas that does not occur with the oil. This process is based upon the fact that when the wet parts of the gas are forced through certain kinds of oil used for absorption, the oil absorbs the light particles of gasoline carried by the gas. These are later recovered from the oil by heating it, the gasoline being driven off and the high-gravity product being condensed. The product of the absorbers is blended with a heavier distillate as is the product from compressing casing-head gas.

During the year 1915 approximately 1,500,000 barrels of gasoline was made by these methods. The value of this gasoline is about \$5,500,000, showing the magnitude of an industry based upon a product that until a short time ago was allowed to waste into the air. This engineering problem is being thoroughly investigated by engineers connected with the Bureau of Mines.

INVESTIGATION OF GAS TRAPS.

The engineering-technology section of the bureau's petroleum division is also investigating the most efficient gas traps for oil wells, with a view to recommending a type for universal adoption.

CHEMICAL-TECHNOLOGY STUDIES OF PETROLEUM.

One of the most important results obtained by the section of chemical technology was the development of the Rittman process which allows a greater percentage of gasoline to be obtained from petroleum than was obtained by other processes. This greater percentage is obtained by subjecting the heavier distillates of petroleum to "cracking," which involves the rearrangement of the molecules of oil by subjecting them to high temperatures and pressures. The Bureau of Mines early realized the importance of this process and as a result of the studies carried on by Dr. Rittman important discoveries were made which bid fair to greatly increase the present supply of gasoline. The Rittman process has proved to be of unquestionable value, although unfortunately not many refineries have installed the necessary equipment for utilizing it.

One of the large fields in refining is that connected with the so-called unsaturated hydrocarbons. Petroleum practice of the past has always dealt with saturated hydrocarbons, namely, those that are not washed out by sulphuric acid. From the chemist's standpoint, the saturated hydrocarbons are inactive, whereas those constituents that are unsaturated are most active chemically, and can be used as a source for building up a great variety of products. The Bureau of Mines desires to extend the scope of such work and recommends this field of research because of its tremendous possibilities.

DEVELOPMENT OF RITTMAN PROCESS.

Another investigation that the bureau desires to continue is the development of the Rittman process. With the present limited appropriations made for petroleum investigations the bureau has not been able to carry on investigations in the field of possible by-products to be made from the cracking of oils. It seems clearly possible to make such by-products as special lubricating oils, aromatic hydrocarbons, and other hydrocarbons that can be used in making drugs and antiseptics. During the past year the Rittman process for making gasoline has passed beyond the experimental state and may be considered a commercial process. During the experiments carried on by the Standard Oil Co. of New Jersey at the Rittman Process Corporation's plant at Pittsburgh, as much as 100 barrels a day was treated in a 11-inch cracking tube, such as is described in Bureau of Mines Technical Paper 161, with a recovery of 40 per cent gasoline having an end point of 165° C., whereas the end point of what is now termed gasoline is nearer 200° C.

The independent refiners have been slow to take up the process, and consequently its development has been retarded. The Bureau of

Mines has been handicapped in that it has not had a plant at which it could carry out tests on a commercial scale. The licensees, as a rule, are not interested in developing the chemical possibilities of the process so much as in obtaining the products they desire. If the maximum progress is to be made in the development of the Rittman process, it seems necessary for the Bureau of Mines to control a plant at which commercial quantities of oil can be treated and to which persons interested in the process can send carload lots of oil to be tested before they invest in the construction of a plant. The labor costs of making such a test should be charged against the applicant for the test.

CONTEMPLATED INVESTIGATIONS.

In the future the bureau hopes to investigate sulphuric acid residues from the refining of petroleum; the possible by-products obtainable from acid sludges and cracked oils; and the internal-combustion engine fuels, which will involve the testing of various grades of such fuels for the purpose of determining relative efficiencies and satisfactory end points and other limits. The bureau also hopes to investigate and prepare specifications for lubricating oils and to standardize, as nearly as possible, the methods of testing petroleum and all its products. The best method of analyzing crude oil must be studied. Gasoline, which is one of the Nation's greatest necessities, is being thoroughly investigated, and thousands of analyses are being made to determine the grades of gasoline being used in different parts of the United States, so as to enable the Bureau of Mines to furnish expert information to State governments and private individuals as to the essential properties of gasoline for various uses.

During the past year the increasing demand for gasoline has been met in four ways: By stimulating production; by the use of a cracking process controlled by the Standard Oil Co.; by increasing the end point from 150° C., the end point of a year ago, to an end point as high as 200° C., which has permitted the use of large quantities of distillate that was formerly considered to be naphtha and kerosene stock; and by blending casing-head gasoline with heavy naphtha or light kerosene stock. It is impossible to count upon an increased production of crude to keep pace with future increases in the demand; likewise it is impossible to expect a much further raising of the end point without a decided change in engine designs, for the reason that the present end point includes most of the products that can be heated up to their vaporizing point without cracking, and if the products are heated above their cracking point objectionable decomposition products are formed. The casing-head gasoline industry will probably never furnish more than 10 per cent of the total

production of gasoline; therefore, the only possible way of keeping pace with this increased consumption of gasoline is by means of cracking the heavier oils. We are to-day using efficiently—that is, for gasoline and lubricating purposes—not more than 30 per cent of our oils. The other 70 per cent is used in competition with coal or exported to foreign countries and is generally sold for less than the cost of production. The gasoline problem before the public to-day can probably be most efficiently solved by cracking this 70 per cent of petroleum, thereby increasing the yield of gasoline from our present oil production by 100 to 200 per cent.

EXTRACTION OF OIL FROM OIL SHALES.

The question is being asked daily what this country is going to do when our petroleum resources are exhausted. We have as yet untouched our great reserves of shales that contain oil. These shales are found in many parts of the United States, and tremendous reserves are known in Colorado, Utah, and Wyoming. There is only one country in the world where oil shales are being utilized for the production of oil—Scotland, where little petroleum occurs and where the demand for petroleum is great. Some of our shales are much richer in oil than are the Scotch shales, and are conservatively estimated to contain many times the amount of oil that has been or will have been produced from all the porous formations in this country.

To obtain the oil from oil shale it is necessary to heat the shale in great retorts. The oil is the result of destructive distillation and is driven off in the form of vapor and is later condensed by cooling. As stated above, this process has never been used in this country because of the lack of necessity, for our oil reserves are great, and it would not be commercially economical to invest money in retorts for distilling from oil shale oil that would have to compete with the crude oil obtained by other methods. But this condition will not last forever. In fact, it is thought that it will be only a very short time until the oil-shale industry will be one of magnitude.

METALLURGICAL INVESTIGATIONS.

GENERAL REMARKS.

For an intelligent understanding of the work and plans of the metallurgical division of the bureau, some knowledge of its history is necessary.

Owing to the lack of any appropriation for the purpose, the bureau did not take up metallurgical studies until 1911, when a modest beginning was made by establishing a laboratory at San Francisco, Cal., chiefly to study the smelter-fume problem, which was then a particularly live issue between a number of the western smelting companies and the neighboring agricultural interests.

A considerable part of the expense of the investigations centering at the San Francisco station has from the beginning been met through cooperative arrangement with the industries and communities most directly interested.

In 1912 Congress gave the bureau its first appropriation of \$50,000, and in later years this has been increased to \$100,000. Federal metallurgical investigations will be further supplemented by the work of the new mining and metallurgical experiment stations recently authorized under the Kern-Foster bill. This bill provides for 10 new stations, not more than three of which are to be established in any one year. Congress has appropriated \$75,000 for the work of the first three stations, which began their investigation in 1916. They are situated at Fairbanks, Alaska; Tucson, Ariz.; and Seattle, Wash.

The first year's work in these new stations, both in mining and metallurgy, is being given chiefly to making a thorough survey of the specific needs of the territory they are to serve, to planning and installing the necessary equipment, and also, what is perhaps most important of all, to bringing the bureau's own personnel into close cooperative acquaintance and contact with the local mining and metallurgical interests so as to insure the intelligent guidance of the bureau's efforts toward the real needs of the community it is to serve.

It was not until 1916 that a distinct division of metallurgy was officially recognized in the bureau, and the scattered pieces of metallurgical work previously undertaken under other divisions was collected under one head to form the nucleus for a more comprehensive and carefully proportioned program.

With this much by way of introduction and explanation as to how the bureau's metallurgical work started, and bearing in mind the unifying process now being brought to bear upon these somewhat

scattered elements, we may turn to a more detailed description of the bureau's work on metallurgical problems during the fiscal year 1916.

GENERAL STUDY OF THE IRON INDUSTRY.

As iron is the most important of all metals, and the one of which this country is the leading producer, with 40 per cent of the world's output to its credit, it may seem that the bureau has given the iron industry scant attention in comparison with other studies. However, this seeming oversight has been due chiefly to a realization of the magnitude of the problems involved and the inadequacy of present appropriations for more than a small start on the work. Furthermore, the iron and steel industry is predominantly in the hands of large industrial interests fully able to work out their own technical problems. Therefore it has been felt that there was not the same urgency in proffering Government assistance in this direction as there has been in the industries representing smaller and more scattered control and individual effort.

Still, even in the iron industry an attempt has been made to pick out the phases in which the public itself was most directly interested and to give those features such attention as the funds and facilities at hand would justify.

As a general basis for public understanding of the iron industry, and to serve in answering many of the questions coming to the bureau from all sides regarding the fundamentals of the industry as a whole, a popular report, entitled "The Story of Iron," has been practically completed. The report aims to cover the general history and evolution of the iron industry. The work is both historical and technical and notes the progressive steps and methods in the development of iron manufacture with special reference to the United States; it also cites various noteworthy discoveries, improvements, and processes.

INVESTIGATION OF SAFETY IN IRON AND STEEL INDUSTRY.

In conformity with the bureau's general policy of promoting safety and welfare in the industries which it touches, it has been making a study of accidents at iron and steel plants with a view to determining the principal causes of such accidents and to developing equipment and practice to lessen them.

Since 1906 a few firms have been actively engaged in accident reduction in their plants, and since 1910 the iron and steel industry has been increasingly putting more and more insistence upon the elimination of dangers and the lessening of the accident risk in the occupation. In 1909-10 the Federal Department of Labor conducted an intensive investigation into the accidents and accident prevention

in the industry. In the main, the report of the investigation was statistical and it showed unmistakably the unusual as well as the ordinary causes of accidents characteristic of blast furnaces and steel works plants. It showed that, in addition to accidents occurring by reason of falls, falling objects, railroad equipment, machinery, hand labor, etc., causes not essentially different from those encountered in other occupations, a considerable proportion of the accidents were caused by strictly occupational hazards, which might be classed as metallurgical accidents. Among these, asphyxiation from gas, burns from cinder and hot metal, and injuries from explosions, breakouts, and slips in the furnaces are typical of the industry.

It was felt that the work peculiarly open to the Bureau of Mines was a study of the metallurgical accidents, because little intensive related study had been devoted to them, and because the other aspects of safety work were being adequately taken care of by State departments, the National Safety Council, and by the efforts of the majority of the large companies. The bureau's work has accordingly been largely concerned with the hazards peculiar to the industry.

The investigation of accidents and of safety devices at blast furnace plants has been finished. It has necessitated much travel—all districts of the country, with the exception of the Colorado district, having been visited at least once—many interviews with men managing, operating, and working at the various blast furnace plants, the compiling of considerable information on operating and constructional features, the study of hundreds of accidents, and the observation of furnace methods and practice. The result of this investigation is represented in four reports submitted for publication. One constitutes a manual of safe practices for furnace foremen and men.^a A second is devoted to asphyxiation from blast-furnace gas.^b A third report treats in detail of blast-furnace breakouts, explosions, and slips.^c

In cooperation with the Pennsylvania department of labor and industry, an investigation has been made of the hazards at furnace plants in that State and the findings, constituting an analysis of the accident risk and methods and means of prevention, have been submitted for publication. The report takes up the entire field of accidents about blast furnaces, and every type of accident is illustrated by the description of an actual occurrence, together with discussion of means best calculated to prevent it.

^a Willcox, F. H., *Safe practice at blast furnaces; a manual for foremen and men*: Tech. Paper 136, Bureau of Mines, 1916, 73 pp., 1 pl., 43 figs.

^b Willcox, F. H., *Asphyxiation from blast-furnace gas*: Tech. Paper 106, Bureau of Mines, 1916, 69 pp., 8 pls., 11 figs.

^c Willcox, F. H., *Blast-furnace breakouts, explosions, and slips, and method of prevention*: Bull. 130, Bureau of Mines. (In press.)

COOPERATION OF UNITED STATES PUBLIC HEALTH SERVICE.

In the safety investigations, as in other divisions of the work elsewhere noted, the bureau has had most important and cordial cooperation from the Public Health Service through the detail of surgeons.

To aid in improving hygienic conditions a plan was formulated for the investigation of the health of workers in the steel and metallurgical plants of the Pittsburgh district. The investigation began April 16, 1915, and terminated March 15, 1916. During that time seven of the largest and most representative steel plants, employing approximately 35,000 men, were visited. A general survey of each plant was first made and data were collected on such subjects as the sanitary condition of the plants as a whole, the character of the work performed by the employees, the amount required, and the conditions under which it was performed as far as each of these might affect the health of steel workers.

A detailed study was then made of each condition observed that might injuriously affect the employees' health, with a view to devising ways and means by which these health hazards could be eliminated or minimized. A report has been prepared for publication.

CORROSION INVESTIGATIONS.**ECONOMIC LOSSES DUE TO CORROSION.**

The corrosion of metals is receiving constantly increasing attention in the metallurgical world on the part of both the manufacturer and the user. For a time we were inclined to believe that modern conditions of service and the demand for production by present-day methods made short life inevitable. Fortunately scattered examples of long-lived products, and the activities of certain investigators and producers, showed that improvement in material was within the bounds of probability. Such improvement has been marked within recent years, and it seems likely that we have by no means reached the limit of advancement in quality. Again, the better understanding of the underlying causes of corrosion is opening new avenues of attack for solution of the problem.

That the rusting of iron and steel is one of our serious economic and conservation problems is now generally admitted. If we assume an average life of steel of 33 years, the depreciation charge of 3 per cent represents in this country a yearly loss of 1,000,000 tons of product, valued at \$30,000,000 to \$40,000,000 for the crude or semifinished material alone, exclusive of correlated fabricating costs. The inevitable rusting of steel may be justly claimed to be the bulwark of the zinc industry, as 60 per cent of the metallic zinc used in this

country is for galvanizing iron and steel articles, representing an annual outlay of \$20,000,000 in an endeavor to protect metals from inevitable decay. Enormous amounts of paint are used in a like endeavor. About 5,000,000 tons of coal is needed in the production of steel to replace the annual waste and 1,000,000 more for the zinc that is annually lost.

No estimate can be made of the value of the brass, bronze, copper, aluminum, nickel, tin, and other metals and alloys used in machine parts, as sheathing, for plating, etc., to protect steel or as a substitute for it in places where it would be used but for its lack of resistance to atmospheric attack. A less definite sum is represented in the capital, labor, and material charges for steel construction of a costly design necessitated by the fact that the corrosion of iron and steel destroys any assurance that we may utilize the full measure of the valuable properties of the metal for an unlimited term of service.

CHARACTER OF BUREAU'S INVESTIGATIONS.

The bureau commenced its systematic studies of corrosion some two years ago, largely on account of the great importance of this subject in mines where pipes, hoisting cables, tracks, and, in fact, well-nigh every piece of underground iron and steel equipment is exposed to unusually severe conditions of rusting.

Extensive investigations in the field have been made to determine the resistance of various grades of material, including protective coatings, under the effect of a wide range of conditions in actual use, but it would be difficult in the present brief compass to give any serviceable idea of this material as a whole.

What will be perhaps even more interesting to the general reader is some idea of the theory that has for a long time dominated the whole field of practical corrosion studies but which the recent investigations of the bureau's staff have shown is inadequate.

NATURE OF CORROSION OF METALS.

The mechanism of corrosion is of extreme importance as underlying any investigation of the problem. Its study involves research of an electrochemical and metallographic nature. This phase of the problem has of late years received the minor share of attention, partly because the commercial interests are primarily concerned with tests that will prove superiority for their particular products and partly because the electrolytic theory of corrosion has been rather generally accepted, and in consequence most investigators have assumed that the scientific aspects of the problem were largely solved.

The electrolytic theory of corrosion is based on simple and well-understood principles; their application to corrosion phenomena in-

volves modifications of detail only. For the generation and continuance of electrolytic action certain factors are essential, namely, two dissimilar electrodes, each of which must be an electrical conductor; an electrolyte in contact with both electrodes; a metallic connection external to the cell; and a depolarizer to insure continuity of electrolytic action. Steel subjected to the weather presents the necessary factors. The heterogeneity of surface, chemical or mechanical, furnishes the electrodes of dissimilar character; the moisture of the air, in its natural state or made more active because of dissolved carbon dioxide, sulphurous, or other gases, is the electrolyte; the electrical circuit is completed by the contact of the electrode spots through the body of the steel; the oxygen of the air reacts with the hydrogen liberated at the cathode surface and removes it by formation of water, thus effecting the necessary depolarization. Electrolytic action must proceed if all factors are present; it will cease if any one is withdrawn. At points where the iron forms the anodic electrode it will go into solution, be converted immediately into the hydrate, and be precipitated upon the surface as such, to subsequently undergo transition to the familiar hydrates and oxides, which are more commonly called rust.

ELECTROLYTIC THEORY UNSATISFACTORY.

One does not have to go far into the study of the corrosion problem to be forced to the conclusion that many of the actual facts observed in service tests are not satisfactorily accounted for by the usual interpretation of the electrolytic theory. For example, iron of highest purity should be most immune from rusting, and such rusting should be even in distribution, with absence of pitting. And yet pure iron does rust, and it does pit as severely as do impure products, especially in some service, as in water pipes.

The work of the Bureau of Mines during the past year has been devoted to the study of the surface influences, particularly of the influence of rust once formed on the progress of further rusting. The findings have received the general approval of authorities on corrosion as explaining many of the hitherto seemingly anomalous observations, and especially the differences observed in the corrosion of pipe as compared with that of iron exposed to the weather, and the more pronounced pitting common to iron pipe.

EFFECT AND KINDS OF RUST.

In brief, the investigation has shown that rust once formed is a factor in the progress of rusting at least equal in effect to those factors universally recognized and is more important than many factors that are subject to dispute because of being attended with

detrimental effects of considerable magnitude. It has been found that there is a reversal of polarity according as the rust is wet or dried. As regards rust freshly formed or wet, the underlying iron is electropositive to the surrounding metal, and consequently goes into solution; dried-out rust, on the other hand, assumes a negative polarity, which tends to reverse in time after the rust is thoroughly wet. This observation has a most vital bearing upon the differences observed in rusting under various conditions, and it accounts satisfactorily for the hitherto enigmatical pitting noticed in water pipes. A most important finding is that fresh wet rust does not act as an electrode, but as a semipermeable diaphragm which makes the underlying iron electropositive, and thus promotes its solution, and that the same effects may be obtained by colloidal or sedimentary substances. This fact is of importance in explaining many effects observed in iron pipe or in iron used under water.

Some interesting work has been done during the past year in the detection of steel scrap in wrought-iron pipe of supposedly genuine character. Methods for such detection have been devised and will be described in a forthcoming report, so that the information may be used by consumers of pipe.

SLAG INVESTIGATIONS.

On account of the growth and expansion of the iron and steel industry and of the industries which depend upon an abundant supply of iron and steel, there arose at the beginning the last century an urgent need for greater economy and output in the operation of the blast furnace. The raw materials for the manufacture of iron must needs remain iron ore and coal; and with the exhaustion of the higher grade ores the industry is becoming confronted with the problem of profitably working leaner ores or ores that present unusual operating difficulties.

In their preliminary survey of the field the metallurgists of the bureau chose as problems demanding scientific investigation the fluidity of blast-furnace slags and its relation to their chemical composition, and the mechanism and physical chemistry of the desulphurization process—two problems intimately associated with the realization of fuel economy and the production of high grade pig iron.

FLUIDITY OF BLAST-FURNACE SLAGS.

The first phase of these investigations, the fluidity of blast-furnace slags, has for many years been the subject of much speculation and theoretical deduction by operating men and scientists. Prior to the present investigations there had been no reliable conclusions drawn

because of the utter lack of experimental data relating to the problem.

The function of the slag in the blast furnace is the elimination of undesirable impurities from the iron ore in such a manner that the resulting mass may be easily flushed from the furnace without interrupting the smelting process. This requires that the slag be sufficiently fluid; that is, possess a low enough viscosity to flow readily from the furnace and that the composition of the slag be such that it can efficiently remove the last traces of impurity from the pig iron. If a slag is so refractory that the blast furnace has to be maintained at an abnormally high temperature, the result is a high consumption of fuel and the possibility of an off-grade product. In addition, the high temperature shortens the life of the furnace. At present the questions involved are answered mainly by rule-of-thumb methods and strict observance of previous practice. It is the aim of the bureau's slag investigations to furnish metallurgists and furnacemen with reliable scientific data to guide them in their choice of operating factors and to make possible the broadening of present furnace practice to take care of the increasing need of working lean and complex ores.

Since the beginning of the investigation in November, 1915, the problem of slag viscosity has been successfully solved, so far as the method, apparatus, and technique of measurement are concerned, by the development of the Feild high-temperature viscosimeter. By means of this apparatus the viscosity of slags is accurately measured up to a temperature of 2,900° F., which is approximately 900° higher than had been reached by previous investigations. The viscosity-temperature relations of numerous typical commercial slags have already been investigated, and similar measurements of synthetic slags are under way to show the effect of the different constituents, particularly of magnesia, alumina, titanium oxide, maganese oxide, and sulphur. As soon as these investigations are completed the study of the desulphurization of pig iron and its relation to the viscosity and composition of slags is to be undertaken.

The electric furnace and apparatus used in these investigations is shown in Plate XIV, A. The furnace and its accessories were constructed in the instrument and machine shops of the bureau.

Numerous slag samples have been collected from blast-furnace plants throughout the country, and these are being investigated with the apparatus with a view to applying the laboratory results to operating practice.

A report on the methods of measurement, the apparatus, and the laboratory technique used in the viscosity investigations has been published by the Bureau of Mines as Technical Paper 157. The re-

sults of further research will be published from time to time as the experimental measurements accumulate.

It is earnestly hoped that the investigations designed to promote efficiency in the iron and steel industry, of which the present study of slags is the beginning, may be carried to their logical conclusion, and that the necessary funds and equipment may be made available for their continuance.

COOPERATIVE INVESTIGATIONS AT SALT LAKE CITY, UTAH.

An important feature of the bureau's work has been the close association in which it has worked with the universities and mining schools of the country. Nowhere is this better illustrated than in the activities of its Salt Lake City station. This city has of late years become an extremely important mining and metallurgical center. Some four years ago the Legislature of Utah granted an appropriation to the State university of \$7,500 a year for metallurgical research especially looking to the improvement of methods for treating the low-grade and complex ores of the State, great deposits of which were then either lying idle and unworkable or were actually going to waste in the mining and treatment of the higher grade ore associated with them.

The university authorities sought the cooperation of the Bureau of Mines in the carrying out of this important research work, and it was finally decided to put one of the bureau's metallurgists in direct charge. A considerable part of the legislative appropriation mentioned above is each year expended on a number of fellowships, the recipients of which work on individual problems under the direction of the bureau's metallurgist. The remainder of the special fund is used for equipment, supplies, traveling expenses, etc. The station has its quarters in the university buildings and has the use of the university's general equipment, such as its library and laboratories. The present is the third year of the cooperation, and in that time the work has developed so favorably that the bureau now has seven representatives at the station as compared with the one originally there.

SUMMARY OF RESULTS.

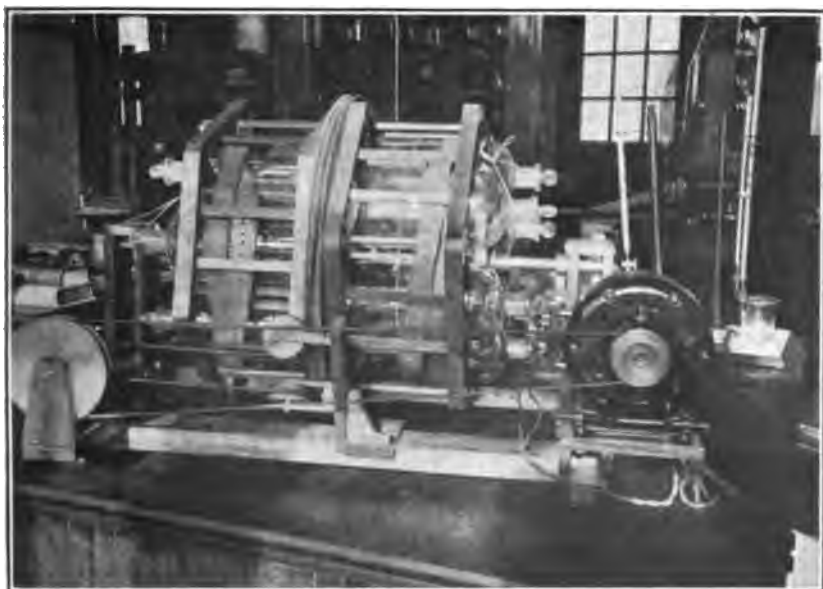
The important results obtained in connection with the various investigations conducted by the station during the year were as follows.

BRINE-LEACHING PROCESS FOR LOW-GRADE AND COMPLEX ORES CONTAINING LEAD.

As stated above, one part of the work of the Salt Lake station is to devise processes for treating low-grade and complex nonferrous



A. ELECTRIC FURNACE AND APPARATUS FOR DETERMINING VISCOSITY OF BLAST-FURNACE SLAGS.



B. PART OF APPARATUS USED FOR LABORATORY EXPERIMENTS ON SMELTER FUME.

ores at a profit. As the values of the metals in the ores are relatively small, the processes must necessarily be simple and inexpensive. For this reason, in devising hydrometallurgical processes, an attempt has been made to use chemicals and such raw materials as are to be found in the State, especially the salt in the Great Salt Lake, and the sulphur gases emanating from the various smelters in the vicinity of Salt Lake City, which are not at present utilized. In attempting to use such reagents it was found that a saturated brine, to which some sulphuric acid had been added, would dissolve the lead from both the carbonate and the sulphate ores of that metal, and that any silver present as the chloride would also be dissolved. Lead carbonate ores from all over the State, and from all over the United States, have been tested and few of them are proved to be not amenable to the process. From the brine the lead can be recovered by precipitation with lime, or by the use of the electric current. The process has proved remarkably simple and the brine used in dissolving the lead from the ore and precipitating it in the metallic form has been reused time and again without deterioration. In fact, the process is proving to be about as simple as any hydrometallurgical process yet devised. It is being further tested with all available samples of ores in order to determine its limitations.

If the process proves to be commercially applicable, great bodies of what is now considered to be waste will soon be regarded as ore.

One great advantage of such a process is that by its use metal can be produced at the mine, which has never been done, except in the cyaniding of gold and silver ores, and to some extent in the leaching of copper ores. As can be readily understood, if the valuable metals of an ore can be extracted at the mine, the cost of hauling the ore to a smelter is eliminated. This of course means cutting down the expense connected with the treatment of the ore, in turn making possible the profitable treatment of a lower grade of rock.

The application of such a process also means the avoidance of waste, for at present ores that can not be treated as a profit are left in the mines, being generally used for filling stopes, and hence ultimately lost.

RECOVERY OF LEAD AND ZINC FROM MIXED SULPHIDES.

As a rule, in treating ores containing a mixture of lead and zinc sulphides for the recovery of their lead or their zinc, either the lead or the zinc is lost. Moreover, the lead smelter penalizes the shipper of an ore if it contains zinc in excess of a certain stipulated per cent, say 10 per cent, and the zinc smelter does the same if the ore contains lead in excess of a certain stipulated per cent. Consequently the separation of the two metals from each other as completely as pos-

sible is important. Much attention has been given to this problem by the metallurgists at the Salt Lake station, and as a result of their work they have devised a process whereby the mixed sulphides, plus a small per cent of common salt, are given a preliminary roast, which converts practically all of the lead to a form soluble in a saturated brine.

The zinc sulphide is untouched by the roast and can often be removed from the remaining iron sulphide by flotation. Certain zinc ores and zinc concentrates obtained as a result of ordinary gravity processes of concentration often contain a considerable amount of lead, for which as a rule nothing is paid, unless the lead-zinc ore is sold to manufacturers of pigment, or contains enough lead and silver to warrant sending the residue from the zinc plant to a lead smelter. Even then, only 60 per cent of the value of the silver in the residue will be paid for at the prevailing market price and only 60 per cent of the lead at 2 cents per pound, whereas the normal market price for metallic lead in pigs is about 4 to 5 cents a pound.

Thus we see that there is need of a process that will save the lead that is now lost altogether and will allow the miner to get a money return for the greater part of this lead. When tried in the laboratory the process described above satisfactorily extracted the lead from the mixed sulphides of lead and zinc.

APPLICATION OF OIL-FLOTATION PROCESS TO ORE SLIMES.

In the concentration of ores by ordinary gravity processes, heavy losses of the metal-bearing contents of the ore often occur, because the necessary crushing and grinding convert some of the minerals into fine slimes which are not recovered in the process of concentration. For this reason an investigation was undertaken that had for its object the recovery of these slimes by the oil-flotation process.

The investigation has shown that it is possible to apply the flotation process to the treatment of some of the lead carbonate ores not amenable to the brine-leaching processes previously described. If silver insoluble in brine is present in noteworthy amounts, the flotation process recovers both the lead and the silver in a high-grade concentrate, which can be sold to the smelters. Furthermore, ores of lead carbonate that contain too many acid-consuming minerals to permit brine leaching are amenable to the flotation process.

Thus it will be noted that as a result of the investigations carried on at the Salt Lake City station during the past year processes have been devised that permit the treatment of various carbonate ores of lead. Their importance as regards the treatment of low-grade lead carbonate ores can not be overestimated.

RECOVERY OF ZINC FROM LOW-GRADE OR COMPLEX ORES.

During the past two years many proposed processes for the treatment of the oxidized ores of zinc have been investigated at the Salt Lake City station. In the treatment of most ores the proposed hydrometallurgical processes have been found to present serious difficulties. On the other hand, zinc sulphide ores are at present being very successfully treated by such processes. Owing to the difficulties encountered in attempting to apply hydrometallurgical processes to oxidized ores of zinc, an igneous concentration process was tested, with encouraging results. The process consists in blowing a blast of air through a mixture of oxidized zinc ore, coke, and limestone. As a result the zinc oxide was drawn off as a fume, which could have been collected in a bag house, or by electrostatic precipitation. The ore from which the zinc has been volatilized forms a slag in the furnace and is tapped out in the usual manner. The commercial application of such a process is also important, as at present there is no process being used that permits the successful concentration of low-grade oxidized ores of zinc. Such ores have been accumulating in all of the important mining districts of the United States, especially in Utah, where there are large deposits of such ores.

SULPHUROUS ACID PROCESS FOR COPPER AND ZINC-LEAD ORES.

During the past fiscal year an investigation has been made to develop a method of utilizing the sulphur gases of the smelters, which are now not only wasted but at many smelters do damage to adjacent vegetation. The sulphur dioxide, which is the most important constituent of these gases, is an excellent solvent for the oxides of zinc and copper. However, in connection with the use of the gas as a leaching agent there are still problems to be solved, among which may be mentioned the obtaining of the gas from ordinary smelter gases in a sufficiently concentrated form and the subsequent precipitation of the metal from the sulphurous acid solution. Both of these problems have already occupied the attention of many investigators, and the former was studied at the Panama-Pacific Exposition laboratories of the bureau, as detailed elsewhere in this report. A commercial method has been worked out for obtaining copper from low-grade oxidized ores of copper by burning elemental sulphur for the production of sulphur gases. The investigation outlined will be continued.

PRODUCTION OF ZINC DUST FROM SOLUTIONS OF ZINC.

Since the advent of the European war zinc dust, which is used in connection with the cyanide process for the precipitation of gold and

silver from cyanide solutions, has been hard to obtain. For this reason an investigation was undertaken which had for its object the preparation of zinc dust from solutions of zinc by an electrolytic method. Promising results were obtained. Moreover, the zinc dust obtained was of higher efficiency than the ordinary zinc dust of commerce.

CONTROL OF ORE SLIMES.

In the crushing and fine grinding of ores in concentration processes slimes are produced, and many of the tanks required for the settling of these slimes cover a large amount of floor space. Consequently any method by which these slimes could be quickly settled, so that the number of tanks or vats required would be decreased, would result in the saving of considerable floor space and cost of tanks and hence permit treating the ore at less cost. A study was undertaken with a view to developing such a method. Encouraging results were obtained. For instance, it was found that if certain mills should adopt the methods developed at the Salt Lake City station the cost of the slime-settling machinery could be reduced by about one-half.

WORK IN SOUTHWESTERN MISSOURI LEAD AND ZINC DISTRICT.

A detailed study of conditions in the lead and zinc mines and mills of the southwestern Missouri district, with especial reference to efficiency and losses in milling, was carried on during the past year. This work was undertaken in cooperation with the State School of Mines of Missouri, and the following features were studied: The character of the ore and gangue and the economic geology of the ore deposits; the efficiency of the milling methods; the possibilities of flotation as applied to the Joplin ores. This study was closely correlated with the work of the bureau's mining division in the district.

Some interesting and valuable data were obtained from the study of the efficiency in the milling practice and from the results of detailed mill tests. The mill tests showed especially where the losses were taking place in the mills and assisted greatly in determining what commercial improvements are possible. The information collected is to be published in a bulletin by the bureau.

COOPERATIVE METALLURGICAL EXHIBIT AND LABORATORY AT PANAMA-PACIFIC EXPOSITION.

Another good illustration of how the Bureau of Mines has endeavored in every way open to it to cooperate with the industry itself and take advantage of each special opportunity presented to bring the fruits of its work home to all the people who may be able to use them is found in the cooperative metallurgical exhibit and

laboratory (Pl. XV) maintained at the Panama-Pacific International Exposition.

The organization of the cooperative metallurgical exhibit resulted from a number of conferences held during the summer of 1914 between representatives of the exposition management and of the Bureau of Mines.

Plans were developed for a group of metallurgical laboratories where serious experimental work could be carried out. However, in view of the high educational mission of the exposition and the unusual opportunity offered by it of presenting to the nontechnical public a panoramic view, as it were, of the highly important but relatively unfamiliar metallurgical processes practiced in the West, a number of features of a general and popular character, such as charts, photographs, and operating models were included. In addition to the equipment for the experimental laboratories a number of full-sized pieces of metallurgical machinery were included in the exhibit.

Owing to the fact that the Government funds available were inadequate to make the project as broad and representative as possible it was decided to seek the cooperation of the mining and smelting companies and the manufacturers of metallurgical equipment, who gave the project hearty and generous support. The cooperation of the mining departments of the neighboring universities was also procured. A fund of about \$15,000 was subscribed in addition to loans of apparatus and equipment representing an investment of at least \$25,000.

The exhibit occupied approximately 7,000 square feet of floor space near the entrance of the Palace of Mines and Metallurgy. The attention of the visitor entering the building was usually arrested by the bright glow from the battery of assay furnaces situated near the railing surrounding the open laboratory space. Located conveniently near these furnaces was a group of crushing, grinding, sampling, and screening apparatus. Just back of the assay furnaces was a large roller agitator for solutions, and laboratory work benches. The equipment formed the so-called hydrometallurgical laboratory for the study of ore treatment by wet processes. During the period of the exposition several investigations were carried out in this laboratory, including studies of the cyanide treatment of complex silver minerals, the behavior of aluminum as a precipitant of gold and silver, and the comparative efficiencies of wet and of dry methods of screening. It will be noted that these investigations were all of a general nature, dealing with questions of fundamental importance to the industry. This same consideration was a fundamental criterion in the selection of problems for study in other sections of the laboratory.

HYDROMETALLURGICAL INVESTIGATIONS.

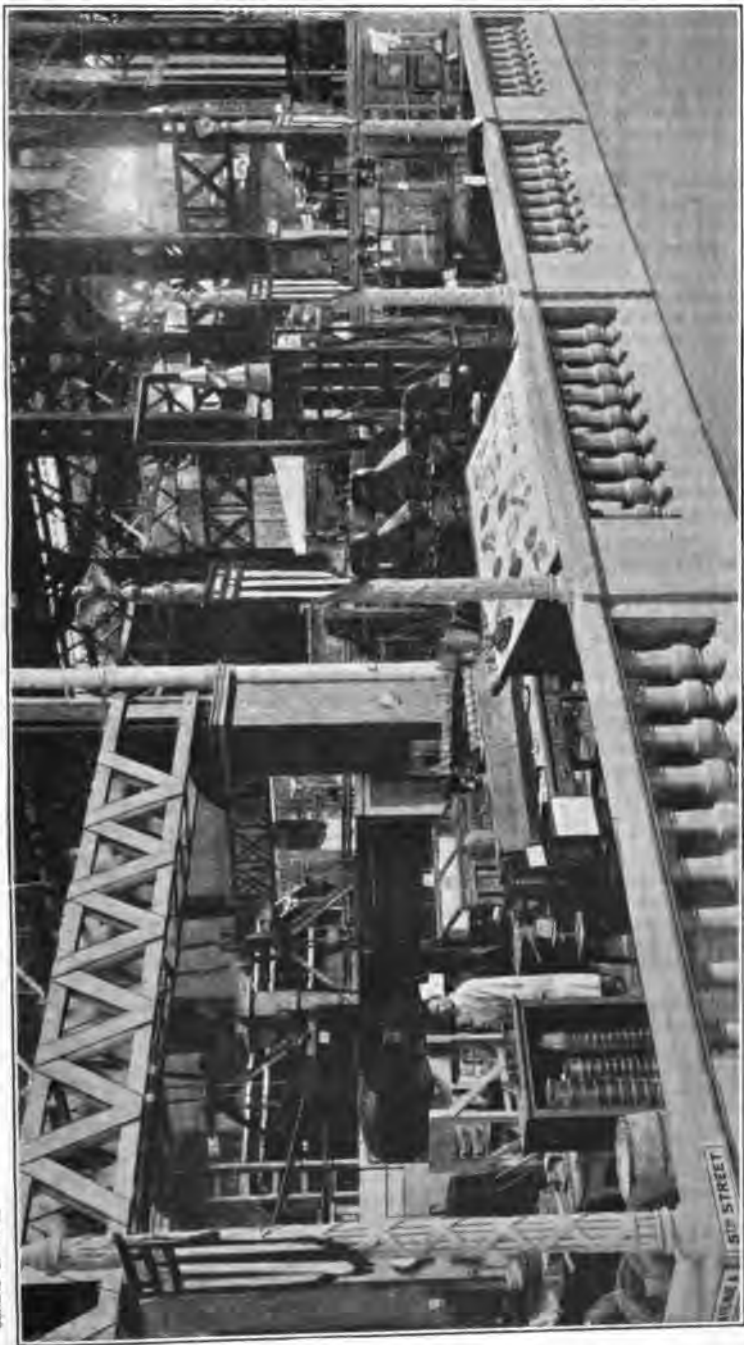
At this point it seems wise to outline the general character of the bureau's hydrometallurgical investigations. These concern the treatment of ores of the precious and base metals by wet methods, as distinguished from pyrometallurgical practice, or the recovery of metals by igneous fusion. The object of the investigations is to lead to a clearer and better understanding of the processes, in order that their operation may be under better control. There are two lines of possible improvement in all such processes, or, for that matter, in any recovery process, either metallurgical or chemical—the percentage of recovery may be improved, or the cost of making a given recovery may be lowered. Either makes possible the economical treatment of lower grade ores—material that remains waste until such a time as it can be profitably treated.

The settling of finely divided suspensions in water, or in solvents that may be used, is of considerable moment in hydrometallurgical work. Such material is referred to in mill parlance as "slime." As there was no sound laboratory method known for ascertaining the settling capacity necessary for a mill handling a particular ore, the development of such a method was taken up. Work upon this problem was carried out both at the exposition and in the field, a considerable number of mining companies cooperating in furnishing data of actual operation. The results of mill operation were found to check reasonably close with those obtained by the laboratory method.

STUDY OF BEHAVIOR OF ALUMINUM IN CYANIDE SOLUTIONS.

The behavior of aluminum in cyanide solutions was investigated, as metallic aluminum is used as a precipitant, replacing zinc in certain cases, and furthermore, aluminum in a soluble form occurs in certain gold and silver ores. On account of the fact that there was little specific knowledge regarding the behavior of aluminum in cyanide solutions, the reactions occurring between metallic aluminum and the caustic alkalies were investigated, and a particular study was made of the ore and pulp from the Goldfield Consolidated mine, Goldfield, Nev., as that ore is one of the best examples of the occurrence of soluble aluminum salts in a precious-metal ore. The aluminum occurs as sulphate, and is the result of the decomposition of various minerals containing aluminum.

This work has been completed, and the chemical reactions taking place under the conditions of mill operation have been established. Space is not available for recording a complete outline of the results, but the more important conclusions reached were that there are two aluminates of calcium formed, namely, a primary aluminate when alumina predominates, and a secondary aluminate when lime is in excess.



PART OF COOPERATIVE METALLURGICAL EXHIBIT AND LABORATORY AT PANAMA-PACIFIC EXPOSITION.

The primary aluminate is soluble, whereas the secondary aluminate is not; therefore, in mill operation, if troublesome insoluble aluminum compounds in the precipitate are to be avoided, the amount of lime used in milling must be restricted to no more than that necessary to form the soluble aluminate. It might be explained that lime is universally used as a source of alkali in cyanide work, and also as an aid in the settling of slime; hence, if aluminum is present, it is important to know the behavior of the calcium aluminates formed in the humid way. Upon the addition of lime during cyanide treatment, first the primary and then the secondary aluminate is formed. Presence of aluminum accounts for the rather high lime consumption in treating such ores. It was found that, contrary to popular supposition, precipitated aluminum salts did not cause cyanide wastes by carrying down with them adsorbed cyanide when the sulphates were present. On the other hand, during precipitation of cyanide solutions with metallic aluminum, under certain circumstances when an aluminum salt is precipitated, cyanide is lost through adsorption. In connection with this investigation, it was found that too long contact of the solution with the precipitant, whether aluminum or zinc, caused a loss of cyanide, probably through decomposition by the nascent hydrogen evolved.

EFFICIENCY OF CRUSHING MACHINES.

In ascertaining the efficiency of crushing machines, as well as in the study and control of all hydrometallurgical processes, small-scale screen-sizing tests are of the greatest value. In order that the work of one mill may be compared with that of another, it is necessary that a uniform screen scale be used by all operators, and, furthermore, that a uniform method of making the sizing test be employed. Unfortunately, in the past much confusion has existed, both as regards the screen scale and the method of making the test, therefore it is often not possible to compare directly the results of different mills. It was with the end in view of establishing a standard method of performing sizing tests, as well as for the purpose of ascertaining the limitation of such a method, that this subject was taken up. In brief, the work so far done demonstrates that satisfactory sizing tests may be made either by hand or with a machine.

It must be emphasized, however, that certain precautions must be observed if consistent results are to be obtained, and, furthermore, it must be expected that considerable time will be consumed in making sizing tests by hand. In the sizing of crushed ores it is essential to remove the finely divided amorphous material by washing prior to screen sizing. If this is done, concordant results can be obtained, but if sizing of the original material without such washing is at-

tempted, the time necessary for the operation is increased by reason of the clogging of the finer screens by the fine material, and, furthermore, the proportion of the coarser sizes is greater, by reason of the fact that the fine clay particles adhere to the coarse sizes. The only way to insure their removal is by preliminary washing, as previously pointed out.

CYANIDATION OF SILVER ORES.

Silver occurs in a great variety of combinations with other elements, so that rather complex and obscure reactions and frequent irregularities happen when silver ores are treated by the cyanide process. Hence, the investigation of the cyanidation of silver ores has been undertaken and considerable progress has been made. The first step has been to study the behavior of various synthetic ores prepared from pure silver minerals. The work so far has been confined to argentite (sulphide of silver), polybasite (sulpho-antimonite of silver and copper), and to pyrargyrite (sulpho-antimonite of silver). These are the more commonly occurring silver minerals.

The work has not proceeded sufficiently far to warrant definite conclusions regarding all the points involved, but it is being continued and detailed reports will later be made. In general it has been found the polybasite is greatly benefited by the presence of a lead salt, as is also argentite. The definite results so far obtained tend to show that the opposite is true as regards pyrargyrite. This is of particular interest, as the addition of a small amount of lead acetate to cyanide liquors is a common practice in mills but is based almost entirely on rule-of-thumb data.

ASSAYING EXHIBIT.

The assaying work conducted in connection with the investigations mentioned above attracted much interest at the exposition, and by means of a graphic chart placed next to the aisle visitors were able to follow the many mysterious and seemingly intricate operations of the "fire assay."

COPPER AND LEAD SMELTING LABORATORY.

Beyond the hydrometallurgical laboratory was a section devoted to the study of problems arising in the smelting of copper and lead ores. The equipment of this laboratory included a multiple-hearth roasting furnace of special design, a tilting furnace, cooling and scrubbing towers for gases, apparatus for cleansing gases of dust and fume by the process of electrical precipitation, and equipment such as pyrometers and draft gages. The investigations in this laboratory which centered around the smelter-fume problem included the distillation of sulphur from pyritic ores in roasting practice and

a critical study of the Thiogen process of sulphur recovery. A detailed discussion of these problems may be found on a later page on smelter-flume problems.

BALANCE ROOM AND MISCELLANEOUS EXHIBITS.

Adjoining this laboratory was the glass-inclosed balance room where the delicate weighing operations involved in assaying and analytical work were carried out. Next to this on the west side of the block was the general display room containing several colored charts illustrating present metallurgical practice in the treatment of copper and lead ores, working models of metallurgical equipment, and photographs and samples of ore and metallurgical products. Adjoining this was a room used as a general office in which was a fairly complete collection of chemical and metallurgical handbooks, as well as files of scientific journals. These books were open to the public for reference at all times and were used by many.

The remainder of the western half of the block was occupied by the metallographic and chemical laboratories which, owing to the delicate nature of the apparatus employed, were surrounded by glass partitions. In these laboratories chemical and microscopic studies supplementing the work in the metallurgical laboratories were made. Original investigations were also conducted there, including a study of the reactions between the sulphides of iron and water vapor, sulphur dioxide, and other gases at elevated temperature; the form in which metal losses occur in slag from copper smelting; and the behavior of chemical absorbents for sulphur dioxide from smelter gases.

On the east side of the block was a group of the standard machinery used for the extraction of gold and silver by the cyanide process, covering nearly every step of practical operation.

At the northeast corner of the block was a model sampling mill.

POPULARITY AND VALUE OF EXHIBIT AS A WHOLE.

The exhibit proved popular, and in addition to gratifying the casual interest of visitors, it was unquestionably of great educational value. Classes from neighboring high schools, normal schools, and universities were conducted through the exhibit by members of the staff, who explained the work in progress in the various laboratories, also the general metallurgical process with which the investigations were concerned. The laboratories were open at all times to visitors who desired an intimate view of the experimental work. In the demonstration work the guiding thought was, first, to set forth the latest developments in metallurgical practice and, second, to show how, in line with the modern trend in all industries, exact and highly

specialized laboratory investigations are leading the way to increased efficiency and economy and, therefore, to a great conservation of both our material and human resources.

One of the most significant things about the exhibit was the broad spirit of cooperation shown by the many collaborators. In organizing the exhibit an earnest effort was made to invite the cooperation of all the important companies in the smelting industry. Naturally, some companies found themselves unable to participate, but all who expressed a desire to do so were included in the plans. It is felt that the precedent established by the cooperation between industrial organizations, universities, and a governmental bureau for a broad public purpose such as that represented by the exposition will prove of much value in the years to come. The occasion of the exposition supplied, as it were, the psychological time for undertaking such a project, and it is indeed gratifying that in addition to much valuable work accomplished at the exposition further results may be expected from the continuation of the work at the two neighboring universities, as told more in detail elsewhere.

In carrying through a project of this sort many difficulties had to be overcome, and the closest sort of cooperation between the officials of the Bureau of Mines and the officials of the exposition was absolutely essential. The success attained bears excellent witness of the effectiveness of this cooperation, and the members of the bureau's staff appreciate highly the cordial spirit encountered in all their dealings with the exposition authorities.

SMELTER-FUME INVESTIGATIONS.

As stated previously, the bureau's work on metallurgy practically commenced with a study of smelter fume, and this subject has continued to receive considerable attention up to the present time.

The attention of a visitor at one of the large copper or lead ore reduction works is often arrested by the tremendous quantity of molten slag that is continually being discharged from trains of slag cars and which, flowing down the sides of the slag dump, suggests lava flowing down the mountain side from a miniature volcano. The tailings from the concentrating mill form an even greater mass of waste material which, accumulating year after year, frequently builds up artificial mounds more than 100 feet high and covering many acres.

Gazing on these huge accumulations of waste material one is prompted to inquire whether they do not still contain material of value, and in passing it may be mentioned that the great improvements made in metallurgical processes in the last few years have made it possible to profitably rework this material at several plants.

The huge fume cloud emitted by the main stack of the smelter, although presenting a striking spectacle as it stretches for miles across the country, is not likely to suggest to the visitor the same possibilities of potential value as the enormous accumulations of waste mentioned. It is hard to realize that the weight of the gases leaving the smelter stack is frequently more than ten times as great as that of the slag that the plant produces, and perhaps three or four times as great as that of the tailings discharged from the concentrating plant.

Although the major part of this gas is air from which the oxygen has been partly removed in the various processes through which it has passed, the gas stream often contains other material that, if it could be collected in serviceable condition, would be worth a veritable king's ransom. The substances in the gas stream may be divided into two classes, namely, minute suspended particles of solid and liquid matter, the so-called dust and fume, which render the stack discharge visible to the eye, and the truly gaseous material which is by itself invisible. The suspended matter is composed of minute particles of ore and other substances from the furnace charge, which are mechanically carried out by the rapidly moving gases, and also of material that has been volatilized by the intense heat of the furnaces and subsequently condensed in the form of liquid or solid particles when the gases have somewhat cooled, just as atmospheric moisture in a warm moist wind from the south condenses into fog when striking colder currents from the north.

As previously indicated, the gaseous part of the stack discharge consists mainly of the gases of the atmosphere with which are mixed water vapor and carbon dioxide from the combustion of fuel in the furnaces, and also sulphur dioxide arising from the burning of the sulphur in the ore by oxygen from the air.

The suspended material in the gas stream from a copper smelter will contain in general not only copper but also lead, zinc, gold, silver, and many other substances associated with the copper in the ore. The value of these metallic constituents discharged with the gases by a large smelting plant may amount to several thousand dollars per day, and it is only within recent years that methods have been developed by which a part if not all of the suspended material can be removed from the gas stream and subjected to special processes for the recovery of the valuable metals.

During the past few years the Bureau of Mines has been able to assist the smelting interests in effecting a recovery of this material formerly wasted. As an illustration may be mentioned the work of the Anaconda Smelter Commission, with which the bureau has been closely associated. This work is designed to improve smoke conditions at the Anaconda smelter in Montana.

THE SULPHUR PROBLEM OF THE SMELTERS.

Although the problem of recovering the valuable solid material from smelter-stack discharge has attracted some public notice, the smelter-fume problem has come to the attention of the public chiefly in connection with the questions of nuisance and damage. Although formerly the suspended dust and fume in the smelter gases were frequently the cause of complaint from the community, this situation has been largely relieved by the developments referred to above, and at the present time the sulphur dioxide in the gases is the principal cause of complaint. Trouble due to this cause has become rather acute in certain districts where agricultural areas lie close to the smelting plants and where prevailing weather conditions are such that sulphur dioxide gas from the smelter in sufficient concentration to injure vegetation frequently comes in contact with the orchards and growing crops.

The tremendous tonnage of sulphide ores of lead, zinc, and copper now smelted in this country makes the absorption and utilization of sulphur from the smelter fume a particularly difficult problem.

Thus far the most generally applicable method of using the sulphur has been the manufacture of sulphuric acid. The two chief limits to its commercial applicability are (1) lack of a local market for the acid, coupled with the difficulty and expense of its transportation to great distances; (2) the great dilution of the sulphur dioxide with air and other gases in most smelters.

The greatest single use for sulphuric acid to-day is in the manufacture of phosphate fertilizer. The proximity of phosphate rock on the one hand, and of a market for the finished superphosphate fertilizer on the other, are usually determining conditions in this matter.

When it is considered that there are many smelters in the country, most of them in the West, each of which daily burns off 250 to 1,000 tons of sulphur from its ores into the atmosphere, and that each ton of sulphur will make 3 tons of concentrated sulphuric acid and 6 of superphosphate fertilizer, the industrial problem of disposing of the sulphur can be better appreciated.

The discovery in Idaho and Montana, within the past few years, of what are probably the most extensive phosphate-rock beds yet found the world over, is likely to have a very important bearing on the problem in the future; but even so, the freight rates on the finished fertilizer to the southern and eastern markets are still practically prohibitive, and the demand for fertilizer on our virgin soils of the West is developing very slowly. A few days' output of the sulphur from a single one of our large western smelters would supply sufficient acid for the present yearly fertilizer demands of the whole Pacific coast. However, the consumption of fertilizer in the West

has more than tripled in the past five years, and eventually the demand for fertilizer will undoubtedly come to be a factor in smelter-smoke treatment.

There is also the chance of so modifying present smelting practice itself that, with the expenditure of little or no extra fuel, part of the sulphur now being burnt up in the furnaces would be distilled off and collected in the unburnt form. Some of the modern developments in smelting practice during the last few years seem to point strongly in this direction, although here again the possible improvements are probably limited to certain branches or departments only of the work, and part of the sulphur would have still to be taken care of by such other methods as already mentioned.

The problem arising from the objections raised by a community or agricultural district to the operation of a smelter in its vicinity, aside from its more distinctly personal aspect, is essentially one of economics. Any factor that tends to make living conditions unpleasant or unhealthful or to decrease the productivity of soil in any district may be regarded as a source of economic loss to a community. On the other hand, a large metallurgical plant employing several thousand workmen is sure to be an important if not the principal factor in the development of the district in which it is situated, and a partial or complete shutdown must invariably result in a loss to a community.

In undertaking to solve this problem on a broad economic basis accurate information is necessary not only as regards the prevalence of smelter gases in the so-called "smoke zone" but also as to the possible injurious effect of these gases upon vegetation of various sorts. During the years of 1913 and 1914, through cooperation with the Selby Smelter Commission, of which Dr. J. A. Holmes, former director of the Bureau of Mines, was, until his death, chairman, the bureau was able to assist in the investigation and adjustment of questions occasioned by the operation of the smelting plant of the Selby Smelting & Lead Co. at Selby, Cal.

WORK OF THE SELBY SMELTER COMMISSION.

The report of the Selby Smelter Commission, which has been published by the Bureau of Mines, describes the extensive investigation made of conditions in the "smoke zone" and adjoining country. This included such work as a study of soil conditions, with especial reference to the presence of material emanating from the smelter and its possible effect on plant growth, a survey of the prevalent plant diseases and of many other factors affecting plant and animal culture. A thorough study of the distribution of smelter gases in the atmosphere of the region, for which it was necessary to develop simple and

accurate methods for the determination of minute quantities of sulphur dioxide in the air, formed an important part of the work. This report also contains the record of an extensive series of experiments to determine both the nature and the economic consequence of the action of dilute sulphur dioxide gas upon growing vegetation.

Still another phase of the work dealt with a study of the metallurgical processes within the plant in the course of which the smelter gases were produced, the purpose being to ascertain what changes in operation could be made to reduce the quantity of fume evolved should the field work show that this was necessary.

As a result of this investigation the protracted litigation between the Selby Smelting & Lead Co. and the citizens of Solano County, Cal., was brought to a close and conditions were outlined by the commission, by the observance of which the smelting company might continue operations without violation of the court decree. In this connection it is interesting to note that the plan adopted by the Selby Commission of basing their findings upon the results of investigations by a corps of skilled experts has come to be generally recognized as the common sense method of approaching the problem. This is evidenced by the fact that the methods of the Selby Commission have since been adopted at several places, notably in the vicinity of Salt Lake City, where one of the large smelting companies is conducting an extensive investigation of the fume situation in the area surrounding the plant.

STUDY OF SULPHUR-RECOVERY PROCESSES.

As previously mentioned in the general discussion of the sulphur problem, several methods have been suggested for the recovery of solid sulphur as a by-product from smelting operations. Although considerable time and effort has been expended in developing some of these processes and in certain cases experimentation undertaken on a large scale and at heavy expense has shown comparatively little progress. The lack of success can be traced to insufficient knowledge concerning the fundamental physical and chemical principles underlying the process used. The mistake has frequently been made of assuming that because a certain qualitative chemical change is known to take place it can without question be made the basis of an industrial process. Thus, one observing year after year the almost ceaseless motion of the ocean waves and the tremendous force which they exert at times is unavoidably impressed by the energy contained in the moving water. The observation of this fact does not, however, show that this energy can be profitably utilized for it may turn out that the cost of constructing and operating apparatus suitable for the conversion of the energy of the

waves into electrical energy, let us say, will be out of all proportion to the value of the electric current generated.

Therefore, in order to determine whether any of the processes already proposed can be successfully operated and be used as the foundation for the development of new processes, much experimental work is necessary. A special laboratory (Pl. XIV, *B*) has been equipped by the bureau for work upon this phase of the smelter-fume problem. Here a sample of ore, the exact chemical composition of which has been determined by careful analysis, can be submitted to a chemical process the conditions of which can be accurately controlled and measured by the operator. These experiments can frequently be made with a few grams (1 ounce equals 28.3 grams) of material, and more useful information can be obtained than if the experiment is carried out with tons of material on an industrial scale involving the expenditure of vastly more time and money.

Other experiments are made with larger quantities of material and with larger equipment, such as a roasting furnace of special design in which 100 to 200 kilos (1 kilo=2.2 pounds) of ore may be treated in 24 hours.

LABORATORY INVESTIGATION OF SULPHUR RECOVERY.

The objects of the experimental work conducted during the past year in this special laboratory were as follows:

1. The direct production of solid sulphur by the modification of smelting operations in common use.
2. The reduction of sulphur dioxide to solid sulphur by the use of solid, liquid, or gaseous fuel.
3. The production of concentrated sulphur dioxide from dilute smelter gases.

As an illustration of the type of problem falling under group 1, the following may be mentioned:

Processes for the treatment of sulphide ores, involving the use of steam to facilitate the recovery of a part of the sulphur in the solid state, have from time to time been proposed and patented. The advantage claimed for the use of steam is based in practically every patent on the assumption that certain chemical reactions take place between the ore and the steam or water vapor. Therefore, in order to judge the possibility of success of any of these processes it is necessary to determine the degree of chemical reaction when water vapor comes in contact with ore within a furnace. If only a slight reaction occurs under the various conditions that can be produced in a furnace, it is obvious that the presence of water vapor is not likely to affect materially the changes taking place in the furnace, and the production of any important quantity of sulphur through its agency

is not to be expected. An accurate knowledge of the nature and extent of the reactions taking place at various temperatures between water vapor and the metallic sulphides in the ore is therefore essential, and this is one of the things that is being carefully studied by the Bureau of Mines.

As part of the work of the Bureau of Mines' laboratory at the Panama-Pacific International Exposition, it was arranged to cooperate with the Thiogen Company in a critical study of a process that it had developed for the recovery of sulphur from smelter gases. This work falls under group 2 given above. A thorough study has been made of the various steps involved in the "wet Thiogen process," and the results of the investigation are now in process of publication as a Bureau of Mines' bulletin.

From work done upon a laboratory scale it is of course impossible to do more than make a preliminary rough estimate of the cost of using the process on an industrial scale. It was possible, however, to demonstrate that there are no insuperable technical difficulties in the way of the successful operation of the process and that under favorable conditions as regards fuel cost, cold-water supply, market for sulphur, etc., it would probably prove commercial.

These illustrations will serve to indicate the type of work which the Bureau of Mines is doing in its efforts to assist in the solution of the sulphur end of the smelter-fume problem. As previously mentioned, no one process can be hoped to afford a complete general solution of this problem. Alteration in economic conditions as well as the extensive changes and improvements in metallurgical practice and the development of new chemical industries will from time to time make it possible to increase the utilization of by-product sulphur from smelting operations. The Bureau of Mines is giving close attention to this subject in order that it may assist such developments in every way possible.

FLOTATION OF ORES.

One of the most striking developments in metallurgical processes in the past few years has been the successful results obtained with oil-flotation methods of recovering metals from their ores, linked with the installation of extensive equipment for utilizing the process. Accordingly, it has seemed fitting to present a short account of the history, significance, and present status of the process.

HISTORY OF FLOTATION.

The flotation of minerals dates back to the accounts of Agricola, who tells us of the virgins who dipped greasy feathers into the stream and drew forth gold. Strangely enough, gold and other minerals of metallic luster tend to stick to a greased surface, whereas

the worthless gangue minerals that accompany them do not. This property of preferential oiling of metallic minerals has been utilized in many ways and has resulted in the modern flotation process.

The earliest account of the use of a flotation method in the United States is that of John Turnbridge, of Newark, N. J. It appears in patent specification No. 207695, granted to him on September 3, 1879. It is a method for recovering precious metals from the jeweler's wash waters and consists in plunging the water through a bath of oil, which collected the mineral particles.

The person who is credited with being the mother of modern ore flotation is Mrs. Carrie Everson, of Chicago, Denver, and California, successively. However, her flotation patents are antedated by not less than nine other patents involving the flotative principle. She proposed to treat ores by pulverizing them and mixing with water and with oil amounting to 5 to 17 per cent of the weight of the ore. The valuable minerals were collected by the oil and skimmed off of the pulp. Since then literally hundreds of United States patents have been granted for various flotation proposals, but the fact remains that a woman was the first person to attempt commercial recovery of valuable minerals from low-grade ores by this process. Since that time various modifications of the process have been made.

In 1906 Sulman and Pickard took out a patent in which they claim the production of an air froth. A slight amount of oil, amounting to 1 pound per ton of ore, was used, being agitated with about 4 tons of water. The oil caused the water to froth and the finely divided particles of valuable mineral were concentrated in the froth floating on top of the pulp. Some previous investigators had proposed air bubbles to assist bulk oil flotation of the minerals but they claimed that their froth consisted almost entirely of air and films of water slightly contaminated with the frothing oil.

This is the basis of modern froth flotation, and it is the only type now considered of much practical importance.

The owners of the Sulman and Pickard patent, the Minerals Separation Co., Limited, are now engaged in litigation with individuals who feel that the patent was antedated by previous knowledge.

America was late in adopting the process and we owe the first successful commercial development of flotation to the various companies operating at Broken Hill, New South Wales, Australia.

However, when American engineers do take up a process, its use spreads rapidly. This is true as regards the flotation process, and as a result practically every mining company in this country which subjects its ores to a preliminary ore-dressing process before shipment, and which has had trouble recovering all the valuable metals in its ore, has experimented with this process in hopes of procuring better extractions.

The first successful application of flotation in the United States that was of any commercial importance was probably that at the plant of the Butte & Superior Copper Co., Butte, Mont., in 1912. At the present time there are probably as many as 200 concentrating mills in the United States using flotation.

VALUE OF THE PROCESS.

The flotation machines used in these mills are of many diverse types and the developments during the past year have been largely improvement of machinery for accomplishing the same effect with less power.

The reason why the flotation process has proven to be such a valuable addition to the former milling and ore-dressing processes is due to the fact that it is adapted to the treatment of finely ground ore pulps; in fact, before this process was developed, the ore slimes formed during crushing in a concentrating mill were a source of serious losses because no machinery had been devised that could save a high proportion of the valuable metals in such fine material.

On this account the practice previous to the introduction of the flotation process was to crush the ore so as to prevent, as far as possible, the production of fine material. However, more or less of the metal contained in all ores subjected to an ore-dressing process was lost during the treatment of the ore, and for that reason nearly every concentrating mill in this country, of any size, has adopted flotation during the past year or two in order to treat its slimes.

WHERE THE PROCESS IS APPLICABLE.

It is not to be inferred that the flotation process can be successfully applied at every mill where values are being lost. As a rule, the process can be successfully applied if the values being lost in the tailings from such processes are in the form of sulphides. However, the values lost in the tailings from such processes are not always sulphides, but many of them constitute oxides. These are not recovered by the ordinary flotation process. For this reason the Bureau of Mines has done much experimental work in connection with this process with a view to devising modifications of the process by the addition of such reagents as will permit the saving of the metals present not only as sulphides but as oxides as well. It has been very successful in its work along this line and has found that the process can be successfully applied to the treatment of oxidized minerals that usually do not have a metallic luster. In this way the effectiveness of the process for the prevention of mineral waste, one of the main objects of all the bureau's work, has been greatly increased.

The flotation process, besides preventing mineral waste at mills using ordinary gravity concentrating processes, can also be applied

to the recovery of metals contained in tailings from former mill treatment of ores, if the tailings have been adequately preserved by impounding, or otherwise.

In this connection it may be stated that the bureau is endeavoring to emphasize the obligation the mining industry owes to the Nation of storing for future treatment those ores and mill products containing metals that can not be successfully recovered at the present time, for, as time goes on, organized study of metallurgical processes will make possible the commercial use of leaner and leaner ores, and thus extend indefinitely the life of our mineral resources.

The flotation process also offers a method of treating ores that could not heretofore be successfully concentrated, owing to the fact that the valuable minerals contained in them are in such small particles that the ore must be ground as fine as flour before the valuable minerals can be liberated, so that when ordinary gravity concentrating processes are used the valuable particles of the ore are lost.

SUMMARY.

In summary it may be said that the flotation process has been successfully applied to the treatment of:

1. Tailings from ordinary gravity concentrating processes, in which the valuable metals previously lost are in the form of sulphides.

2. Tailings in which the valuable minerals are oxidized, the use of sulphidizing reagents being generally necessary in order to superficially convert the metallic particles into sulphides.

3. The concentration of ores that contain the valuable minerals in such form as to make recovery by ordinary gravity processes impracticable.

Probably no other ore-dressing or metallurgical process devised in recent years has so materially aided in the prevention of mineral waste as has the flotation process. It is safe to predict that in time the scope of the process will be greatly extended and that methods will be developed for treating all types of ores by it.

TECHNICAL-SERVICE WORK AT PITTSBURGH EXPERIMENT STATION.

Closely related to practically all the activities of the Bureau of Mines is the technical-service work done at the bureau's Pittsburgh experiment station. This comprises the computing and compiling of experimental data, the drafting and designing of experimental equipment, and the preparation of photographs and motion pictures relating to the bureau's investigations. This service aids in the interpretation and presentation of the results of the bureau's investigations, thus performing a necessary fundamental part in the study of problems relating to the mineral industry. During the past year the technical-service work included:

1. Computing the results of field and laboratory tests and making reports therefrom; determining physical and chemical laws from observations made and expressing them in the form of mathematical equations; and compiling conversion tables and making land surveys.

2. Designing apparatus and equipment for experimental work; making mine-disaster maps; engrossing mine-rescue and first-aid certificates; and making illustrations for bulletins and reports.

3. Taking photographs to show: Details and results of tests, details of apparatus, progress of buildings, approved first-aid methods and bandages; making lantern slides and enlargements.

4. Producing new motion pictures; repairing and replacing worn film; duplicating films when desirable, in order to supply the demand; loaning motion pictures and other negatives for the purpose of supplying duplicate pictures to mining companies for educational work; revising films in order to improve and get them up to date. Four reels, in duplicate, illustrating the work of the bureau, have been made and assembled. About 65,000 feet of film is now available for exhibition by bureau employees, an increase during the current year of 25,000 feet. The difficulties encountered by the photographer in making mine-safety pictures underground are shown in Plate XVI, in which the photographer is seen in one of the large copper mines in Arizona at work, with his equipment, including arc lights sufficient to give 40,000 candlepower.

Heretofore the work of the bureau has been considerably interfered with by the necessity for having individual investigators show parties of visitors their work. This difficulty has been obviated by having an engineer familiar with the work of the Pittsburgh station act as an official guide. He shows the visitors the routine work of the station, without interrupting it, usually ending his demonstration with the motion picture "The Work of the Bureau."



BUREAU OF MINES OPERATOR TAKING MOTION PICTURES IN ARIZONA MINE.

NEW OFFICE BUILDINGS AT PITTSBURGH.

The new offices and laboratories for the bureau's Pittsburgh experiment station, contract for which was let in the spring of 1915, have been partly completed and the main offices are now occupied.

These buildings stand on land acquired by the Government, an admirable site adjacent to the grounds of the Carnegie Institute of Technology and near the Carnegie Institute, the Carnegie Library, and the University of Pittsburgh. The tract fronts to the north on Forbes Street, and is immediately east of the Forbes Street Bridge over the deep cut of the Pittsburgh Junction Railroad, which bounds the property on the west.

The main building is three stories high, has a frontage of 332 feet, and is flanked at either end with two-story wings extending back from Forbes Street 211 feet.

The central part of the building, which is 56 feet by 210 feet, will contain quarters for the administration offices and the mining-engineering division and also a lecture room seating 300 persons. The west wing, 48 by 211 feet, will house the mechanical laboratory. The east wing, of the same size, will accommodate the chemical laboratory. In general the interior finish is substantial but severely plain.

In the rear of the main building will be the power plant and the metallurgical and fuel-testing laboratories in a building 55 feet by 220 feet, built of reinforced concrete and placed on a lower level, so as to be invisible from Forbes Street. In this building are an engine house 55 feet by 110 feet, a fuel-testing laboratory 55 feet by 70 feet, and a metallurgical testing laboratory 55 feet by 70 feet. An extension of this building is contemplated to provide a boiler house 55 feet by 110 feet and a 50-foot extension to the fuel-testing laboratory. The general appearance of the buildings is shown in figure 9.

BUREAU OF MINES EXHIBIT ON GOVERNMENT SAFETY-FIRST TRAIN.

During the months of May to August, 1916, a traveling exhibit showing apparatus, devices, and methods recommended by the Government for preserving the health and safety of the people of the United States, visited 87 cities and towns in 16 different States, covering territory all the way from Philadelphia, Pa., to Denver, Colo. The train was organized by the Federal departments and bureaus concerned in the work mentioned, and the 12 steel cars comprising the train were furnished through the cooperation of Mr. Daniel Willard, president of the Baltimore & Ohio Railroad. The train was furnished transportation by the Baltimore & Ohio system, the Missouri, Kansas & Texas system, and the Union Pacific system.

The traveling exhibit was an effort to give the public a clear idea of the humanitarian work being accomplished by the Government. Such an exhibit has never before been witnessed by the people. Those who have been fortunate enough to be able to visit Washington and spend a few days in learning about the Government's activities have been privileged to see a number of these exhibits at various times, but never before has practically the entire safety-first work of the Government been assembled as on this train. That the people were surprised at the extent of the Government's interest in their behalf was manifest every day the train was on its itinerary. It became a common expression that no one before had had an adequate idea of the practical work the Federal Government was doing along these lines.

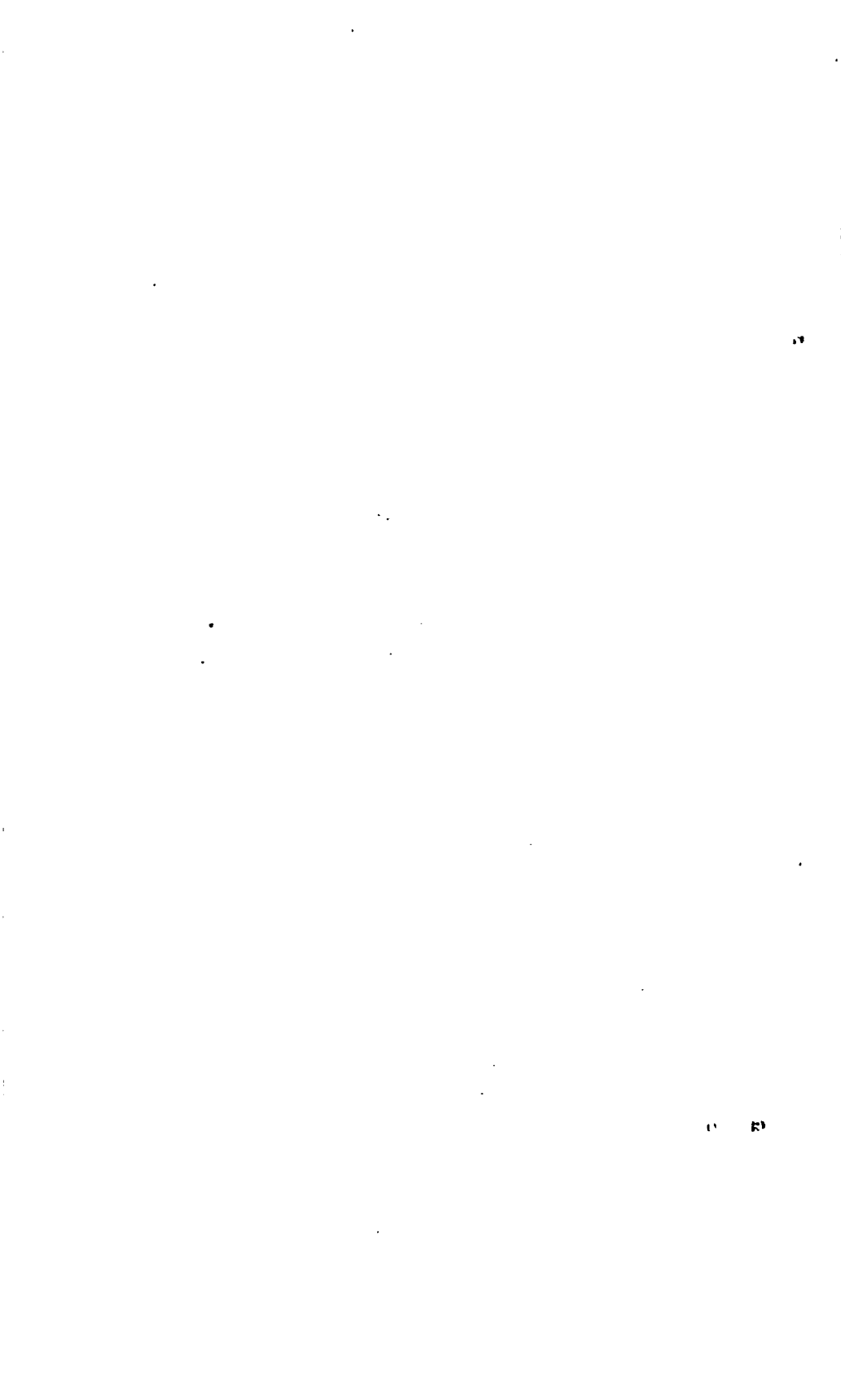
The Government issues millions of publications each year on the subjects depicted by the train, but these unfortunately do not lend themselves to such graphic presentation as the actual apparatus used in these endeavors. It is a safe assumption that many of the visitors left the train better American citizens and with a much clearer view of the helpfulness of the Federal Government to the people.

The Hon. Franklin K. Lane, Secretary of the Interior, was the originator of the plan of having a railway train take to the doorsteps of the people an exhibit of the work of the Government. Secretary Lane not only started the movement but organized the necessary machinery for the actual accomplishment. The details of bringing the various departments and bureaus together were assigned to Mr. Van. H. Manning, director of the Bureau of Mines, who was designated as the executive officer in charge of the train.

With few exceptions all of the material exhibited by the Bureau of Mines on the safety-first train had seen actual service. In the car



NEW BUILDING OF THE PITTSBURGH EXPERIMENT STATION.



devoted to the work of the Bureau of Mines the exhibit which attracted the most attention represented five men equipped with different types of oxygen-breathing rescue apparatus used by the bureau's rescue crews at disasters following explosions.

Much interest was manifested in the cage of life-saving canary birds. The canary is very sensitive to a deadly gas (carbon monoxide) found in mines after fires and explosions, and for this reason is used by rescue crews and miners in testing mine air. When the bird shows signs of distress the men know that it is time to put on their rescue apparatus. The rescuers carry with them an inclosed cage and a supply of oxygen. The asphyxiated bird is placed in this cage and the oxygen turned on, with the result that the bird generally recovers.

Another exhibit was a portable mine-rescue telephone for men wearing oxygen-breathing rescue apparatus and consequently unable to talk distinctly. This telephone has a transmitter that is strapped over the vocal cords and enables a rescuer wearing breathing apparatus to maintain communication with the fresh-air base for a distance of about one-quarter of a mile.

In reviving a person overcome by mine gases, it is sometimes advantageous to administer pure oxygen. The device used by the bureau for this purpose, consisting of an oxygen tank, a reducing valve for regulating the flow of oxygen, tubes, and a face mask, with an outlet valve for the breath, was shown.

Photographs of the rock-dust barriers devised by bureau engineers for checking explosions in coal mines were displayed, and a working model of the trough rock-dust barrier was shown.

A device for sampling mine air for rock dust, used by bureau engineers in investigating the effects of siliceous dust in causing pulmonary disease among miners, was shown.

A photograph of an explosion-proof motor approved by the bureau as permissible for use in gaseous mines was exhibited.

Permissible electric safety lamps and permissible flame safety lamps were shown. One exhibit was a box for testing safety lamps before taking them into a mine, the lamp being tested by filling the box with an explosive gas and then thrusting the lighted lamp into it.

Devices exhibited to insure the safety of miners in handling explosives consisted of a 5-pound powder jack or metal waterproof box for carrying powder into a mine, and a safety box for carrying the detonators used in blasting. In this box each detonator is held separately and only one can be removed conveniently at a time. The detonators can not fall out if the box should be overturned.

There was also exhibited an apparatus to insure safety in firing shots by electricity in mines.

PUBLICATIONS ISSUED BY THE BUREAU OF MINES.

The Bureau of Mines publishes three classes of reports—bulletins, technical papers, and miners' circulars. In addition it issues various lists and schedules, a monthly statement of fatalities in coal mines, and the annual report of the director.

Most of the bulletins present in detail the results of technical and scientific investigations, and therefore are of interest chiefly to engineers, chemists, mine officials, and other persons familiar with the subjects discussed. The technical papers are shorter and less formal than the bulletins and contain preliminary statements of the results of the larger investigations or describe the shorter investigations incidental to a larger one. The miners' circulars deal with topics relating to accident prevention and rescue and first-aid methods, the safeguarding of health, and other matters that directly concern the workers in mines, mills, and metallurgical plants. These circulars are written in simple, nontechnical English, and are printed in much larger editions than are the bulletins and technical papers.

The bureau has already published approximately 100 bulletins, 150 technical papers, and 20 miners' circulars. The following classification gives an idea of the investigations so far represented in the publications of the bureau:

- Publications on the utilization of coal and lignite.

- Publications on the composition of coal.

- Publications on the technology of petroleum and natural gas.

- Publications relating to mining laws.

- Publications on mineral technology.

- Publications on metallurgy.

- Publications on coal mining.

- Publications on metal mining.

- Publications on accident statistics.

- Publications on investigations of mine electrical equipment.

- Publications on investigations of explosives.

The bureau issues a complete list of all its publications, showing those available for free distribution on application to the director and those obtainable from the Superintendent of Documents, Governing Printing Office, on payment of the cost of printing. Interested persons should apply to the Director, Bureau of Mines, Washington, D. C., for a copy of the latest list.

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